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A DETAILED DESIGN/COST STUDY TO ESTABLISH A ROCK BOLTING FIELD DEMONSTRATION PROGRAMME FOR THE SYDNEY COALFIELD, NOVA SCOTIA.

CONTRACT NO. 23440-9-9011/01-SQ

PROJECT NO. 5666

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REPORT TO

DEPARTMENT OF SUPPLY AND SERVICES CANADA

ON

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SUMMARY

The CANMET Cape Breton Coal Research Laboratory retained Jacques, Whitford and Associates Limited of Dartmouth, N.S. through Supply and Services Canada to prepare a detailed design/cost study to establish a rock bolting demonstration program for implementation in the Sydney Coalfield. This report presents details of that study.

Site visits to the three operating collieries in the Sydney Coalfield (Phalen, Lingan, and Prince), and discussions with Headquarters Engineering Staff at the Cape Breton Development Corporation offices in Glace Bay helped to identify potential trial sites and to determine the parameters required to design both the rock bolting pattern and the system to be used in implementing it. Discussions with Labour Canada staff and a review of current safety regulations established a framework within which the trials could be undertaken to conform to existing safety and health regulations.

Discussions with CBDC staff, colliery engineers and site visits to each of the mines helped to identify potential trial sites. Details of mine infrastructure, geology and geotechnical data, and in-situ stress were obtained to assist in the design of rock bolting patterns.

Past work on the design of rock bolting trials was reviewed, with particular emphasis on UK experience; current UK practise, developed in the past few years, is based on extensive study of American, Australian and European methods and as such is an important resource on this subject. The staged approach of implementing rock bolt support advocated by authorities in the UK was adopted and a full set of guidelines for implementing the rock bolting trial were developed.

A cost study was undertaken to determine the effects of rock bolting on mine economics. As expected, rock bolting as a supplementary support with no increase in arch spacing produced an increase in driveage costs. Rock bolting with increased arch spacings or as a sole means of support produced direct cost benefits.

SOMMAIRE

CANMET Cape Breton Coal Research Laboratory à contracté Jacques, Whitford & Associates Limited de Dartmouth, N.E. par l'entremise d'Apprivoisionnement et Services Canada pour préparer une étude détaillé pour établir un programme de démonstration de boulonnage à Sydney Coalfield. Ce rapport présente les détails de cette étude.

Des visites des trois houillères à Sydney Coalfield (Phalen, Lingan, et Prince) et des discussions avec des ingénieurs du siège principal du bureau de Cape Breton Development Corporation à Glace Bay, NE ont aidé a identifier des sites potentiel d'essais et a déterminer les paramètres requis pour établir un système de boulonnage a être utilisé.

Des discussions avec des membres de Travaux Canada et une revue des régulations de sécurité courrantes ont établi un encadrement dans lequel les essais pourrons être entrepris pour conformer aux régulations déjà existantes de santé et sécurité.

Des discussions avec le personnel de CBDC et les ingénieurs des houillères, ainsi que des visites de sites à chacune des mines, ont aidé a identifier des sites potentiel d'essais. Des détails de l'infrastructure de la mine, des données géologiques et géotechniques, et le stress ont été obtenu pour aider a établir un système de boulonnage.

Des travaux déjà entreprient sur ce sujet en Angleterre ont été examiné avec une attention particullère. De récentes méthodes développés durant les denières années sont basées sur une étude approfondi des méthodes américaines, australiennes et européennes. Les étapes approchées pour établir un système de boulonnage recommendé par les authorités d'Angleterre ont été adoptés et des procédure complètes pour implimenter les essais du système de boulonnage ont été développés.

Une étude des coûts a été entreprise pour déterminer les effets économiques des mines. Comme expecté, utiliser le système de boulonnage comme support supplémentaire sans augmenter l'espacement de l'arc produira une augmentation dans le coût de développement de la galerie. Utiliser le boulonnage avec une augmentation de l'espace dans l'arc, ou par seul moyen de support produira des bénifits direct.

Table of Contents

1.0 INTRODUCTION 1

1.1 Background 1

1.2 Statement of Work 1

2.0 ROCK BOLTING IN THE SYDNEY COALFIELD 3

2.1 Current Practise 3

2.2 Future Use 3

2.3 Discussion 4

3.0 DOCUMENTATION OF REGULATORY REQUIREMENTS 5

3.1 Regulatory Requirements 5

3.2 Discussion 8

4.0 IDENTIFICATION OF TRIAL SITES 9

4.1 Selection Process 9

4.1.1 Initial Consultation 9

4.1.2 Preliminary Site Selection 9

4.1.3 Site Visits/Final Selection 9

4.2 Lingan Colliery 9

4.3 Phalen Colliery 9

4.4 Prince Colliery 10

4.5 Documentation of Site Conditions 10

4.5.1 Mining Environment 10

4.5.2 Geology 10

4.5.3 In-Situ Stress 11

4.5.3.1 Lingan Colliery 11

4.5.3.2 Phalen Colliery 11

4.5.3.3 Prince Colliery 11

4.5.4 Geotechnical Conditions 12

4.5.4.1 Lingan Colliery 12

4.5.4.2 Phalen Colliery 12

4.5.4.3 Prince Colliery 12

4.6 Discussion 13

5.0 DETAILED DESIGN OF A ROCK BOLTING TRIAL 15

5.1 Design Strategy 15

5.2 Support Requirement Assessment and Pattern Design 16

5.2.1 Assessment of Support Requirements 16

5.2.1.1 Analytical Methods 16
5.2.1.2 Empirical Methods 17
5.2.2 Rock Bolt Pattern Design 18
5.2.2.1 Analytical Methods 18
5.2.2.2 Empirical Methods 20
5.3 Selection of Equipment 21
5.3.1 Drills, Rods and Bits 22
5.3.2 Bolts, Plates, Nuts and Washers 23
5.3.2.1 Selection of Bolt Type 23
5.3.2.2 Plates, Nuts and Washers 25
5.3.3 Resins or Resin Substitutes 25
5.3.4 Additional Roof Support Components 26
5.4 Installation Procedures 27
5.5 Monitoring and Analysis 28
5.5.1 Technical Monitoring 28
5.5.2 Geotechnical Monitoring 29
5.5.2.1 Mapping 30
5.5.2.2 Core Drilling and Core Logging 30
5.5.2.3 Rock Properties Testing 30
5.5.2.4 Anchorage Testing 31
5.5.2.5 Extensometers 31
5.5.2.6 Load Cells 32
5.5.2.7 Closure Monitoring 32
5.5.2.8 Monitoring Frequency 32
5.5.3 Assessment and Adjustment 33
5.6 Training Programmes 33
5.6.1 Form of Training 33
5.6.2 Training the Workforce 34
5.6.3 Training of Management/Supervisors 35
5.7 Discussion 35
5.7.1 Support Assessment and Pattern Design 35
5.7.2 Rock Bolting Equipment and Supplies 36
5.7.3 Monitoring and Analysis 38
5.7.4 Training 38
6.0 IMPLEMENTATION OF ROCK BOLTING TRIALS 39
6.1 Review of Investigations 39

6.2 Development of an Implementation Strategy 39
6.3 Organization and Responsibilities 41
6.4 Preferred Trial Sites 41
7.0 ROCK BOLTING SYSTEM COST STUDY 43
7.1 Introduction 43
7.2 Costs of Current Support Technique 43
7.3 Costs of Recommended Support Techniques 44
7.3.1. Capital Investment Costs 45
7.3.2 Consumables Costs 45
7.3.3 Manpower Costs 46
7.4 Impact of Rock Bolting on Current Costs 47
7.4.1 Driveage Rates 48
7.4.2 Support Density 48
7.4.3 Time Available for Bolting 48
7.4.5 Cost Variations 50
7.4.6 Economic Viability of Rock Bolting 50
7.5 Summary 51
8.0 CONCLUSIONS 52
9.0 RECOMMENDATIONS 54
10.0 BIBLIOGRAPHY 56

LIST OF TABLES

4.1 Summary of Environment at Each Mine Site
4.2 Stresses Measured at Various Mines in the Sydney Coalfield
7.1 Summary of Drivage Costs
7.2 Savings (Losses) Due to Rockbolting at Prince Mine
7.3 Savings (Losses) Due to Rockbolting at Phalen Mine
7.4 Savings (Losses) Due to Rockbolting at Lingan Mine

LIST OF FIGURES

4.1 Roof Strata Types of the Sydney Coalfield
4.2 Roadway Roof Conditions Caused by the Effects of Lateral Stress
5.1 French Bolting Guidelines
5.2 Untensioned Grouted Dowel

5.3 Single Wire Split Plug Extensometer

5.4 Minimum Recommended Array of Closure Points

5.5 Tensioned, Fully Bonded Resin Rebar Bolt

LIST OF APPENDICES

Appendix 1 Sample Rockbolting Calculation

Appendix 2 Training Programmes

Appendix 3 Detailed Cost Study

1.0 INTRODUCTION

1.1 Background

The Cape Breton Development Corporation (CBDC) and Cape Breton Coal Research Laboratory (CBCRL) share considerable interest in the possible use of rock bolts as a method of supplementary support in coal mine roadways. The use of rock bolts offers several advantages; reductions in the number and section weight of steel arches installed to support development and production roadways, improvements in strata control and economies in development costs. Ultimately, the use of rock bolts as the sole means of support in favourable geological conditions is foreseen if the field trials indicate that they are a viable alternative to steel arches.

Rock bolting is an accepted method of ground stabilization in hard rock mines, and in the coal mines of the United States, Australia and Europe. Western Canadian coal mines using the room-and-pillar method of extraction have also used rock bolts as the sole means of support, although most of these mines are now closed. Rock bolting was widespread in the workings of the Sydney Coalfield, but as longwall replaced room-and-pillar as the dominant extraction method, the practise has fallen into disuse as a systematic method of support.

Acting at the request of CBCRL, Supply and Services Canada retained Jacques, Whitford and Associates to conduct a detailed design/cost study into the feasibility of a rock bolt trial in the Sydney Coalfield.

1.2 Statement of Work

The statement of work, as outlined in the RFP, is as follows;

1. The successful contractor will:

    a) Assess and document CBDC's requirements regarding both their current and potential use of rock bolting techniques in the Sydney Coalfield.

    b) In conjunction with CBDC, identify potential sites where field demonstrations could be undertaken in the Sydney Coalfield operations. Potential sites should be identified at each of the three mines currently operating: Prince, Lingan and Phalen.

    c) Undertake a detailed design/cost study for a rock bolting field demonstration trial at each of the sites identified in (b). Consideration should be given to the use of all types of rock bolts (i.e. resin, friction and mechanical).

    d) Outline in detail the program methodology and requirements necessary to undertake a field demonstration trial at each of the sites identified in (b). Special

consideration should be given to the geotechnical monitoring programs necessary to evaluate the success/failure of the bolting trials as a secondary/primary support method.

e) Undertake detailed discussions with CBDC and Labour Canada to identify and document all the regulatory requirements necessary to implement a field trial.

2. Include all of the above in a final report to be provided to the Scientific Authority by March 1, 1990.

The following report details the results of the investigations conducted under the terms of the contract.

2.0 ROCK BOLTING IN THE SYDNEY COALFIELD

2.1 Current Practise

The current and potential requirements for rock bolting in the Sydney Coalfield were discussed with the Scientific Authority and CBDC engineers at both Headquarters and Colliery level. The current use of rock bolts as a method of ground control is not widespread within CBDC mines. It is restricted to support of the longwall face line prior to powered support salvage and remedial bolting on a semi-systematic basis at face ends. Trials have recently been conducted using cable slings at Phalen Colliery.

At all three operating collieries, Prince, Lingan and Phalen, systematic bolting is used during face salvages. Swellex rock bolts are installed with wire mesh screens in the coal face roof over the last few cuts of the shearer. This allows the powered supports to be withdrawn safely whilst maintaining the faceline open for ventilation.

In the 7 West Bottom Level at Lingan Colliery, which was driven as an advance heading, Swellex rock bolts were installed in the strata overlying the coal on the face side rib to prevent deterioration of the packing lip. Also at Lingan Colliery, the material level brushing lips are bolted using 1.8 m. Swellex bolts to maintain the strata prior to brushing for the arch supports behind the face-line. These bolts are removed with the brushed stone except for one row adjacent to the pack.

In January 1990 trials with cable slings were initiated in the 2 West Top and Bottom levels of Phalen Colliery. However at the time of writing (June, 1990), the trial sections were being worked through and there was insufficient data for any interpretation of their effectiveness. Also at Phalen Colliery some trials were conducted with Swellex bolts as an additional support in the 4 East Top level. Although these appeared effective in reducing crown beam deformation in the gateroads, they were not installed systematically. They continue to be used to alleviate heavy roof conditions in isolated areas.

Although these examples represent the main usage of rock bolts within the CBDC mines, Swellex or conventional tensioned resin rebar type bolts are used intermittently at junctions or at face-ends to control strata movement throughout the coalfield.

2.2 Future Use

All participants in the discussions expressed interest in increasing the use of rock bolts, both as a supplementary support in gateroads and other driveages and as a potential replacement for standing steel supports in appropriate geological conditions. There was general agreement that the use of rock bolts would improve strata control, reduce costs and improve development rates and productivity. One requirement stipulated during the discussions was that any trial of a rock bolting system should not interfere with current advance rates and productivity.

There was also a general agreement that the introduction of rock bolts must be carefully planned and implemented. The best available design methods should be sought out and used, and that every precaution against failure should be taken at the trial stage.

The remainder of this report addresses the design and implementation of bolting trials in development roadways and gateroads. The use of intermittent or spot bolting, bolting of junctions and the use of bolts during face salvages are outside the scope of the program and have not been considered.

2.3 Discussion

Rock bolting as a method of longwall gateroad support has achieved tremendous success in France, Germany and, most recently, in the U.K. As with all new technology, training, familiarization and development of optimal work place performance takes time. Even the replacement of one type of machine by another, or rigid arches by yielding arches for example, causes some disruption.

In the case of a rock bolting trial, when rock bolts are installed in addition to conventional support, the additional time, material and manpower required, at least on the early part of the learning curve, is bound to reduce driveage rates and productivity. This has been confirmed by authorities both in the US (DeMarco, 1990) and in the UK (Newson, 1990, Daws, 1990). However, the extent and duration of disruption to normal production can be reduced by the implementation of comprehensive training programmes prior to the field trials.

3.0 DOCUMENTATION OF REGULATORY REQUIREMENTS

Introduction of rock bolting as a systematic method of support poses significant problems if the regulations do not address its use directly. This is often the case in the traditional coal mining nations where workers are more used to setting steel arches to support roadways. In order to introduce the new technology and support methods, exemptions have to be sought from current regulations, requiring extensive involvement of the regulatory authorities.

Responsibility for Safety and Health regulation of the Cape Breton coal mines was assumed from the Province of Nova Scotia by Labour Canada in 1967. This followed the withdrawal of the Dominion Steel and Coal Company from coal mining in the area, and the establishment of the Cape Breton Development Corporation (a Federal Crown Corporation) as owners and operators of the collieries.

The Federal regulators assumed both Provincial legislation and some Provincial employees in the early stages of the transfer of authority. The provincial legislation was outdated and required significant changes to bring it into line with modern mining methods. Updating the legislation was a slow process, complicated by Commissions of Public Inquiry set up after serious mine fires and a major change in CBDC's mandate; as a result of the international energy crisis in 1972, plans were made to expand operations instead of closing the coalfield.

Collieries operated by CBDC are regulated by Part II of the Canada Labour Code, "Coal Mines (CBDC) Occupational Safety and Health Regulations" (Department of Labour, 1990), a specially written chapter of the Canadian Labour Code, enforced by the Canadian Department of Labour. These regulations deal with strata control and the support of underground openings in only the most general fashion.

3.1 Regulatory Requirements

Prior to the most recent amendments, mining regulations left the design and implementation of strata control measures to the colliery operators, subject to the sanction that Labour Canada Inspectors could demand changes to unsafe working practises. Amended regulations came into effect on February 15, 1990. These demand that new methods and technologies be referred to the Coal Mine Safety Commission, which consists of representatives of Labour Canada, CBDC and colliery employees. Discussions with the Labour Canada District Manager in Sydney lead us to believe that the rock bolting trial would fall into this category.

All strata control methods are dealt with in general form in paragraph 159 of the regulations, as follows:

159. (1) The employer shall prepare a plan of strata control for any proposed underground workings of a coal mine that is designed to prevent the collapse of the roof and sides of those workings.

(2) A copy of the plan referred to in subsection (1) shall be posted at every meeting station in such a manner that it is readily available for inspection by employees.

New technology, methods or equipment is dealt with in paragraph 158, as follows (emphasis by JWA):

158 (1) **The employer shall submit to the Coal Mine Safety Commission for approval a notice in writing of the employer's intention to carry out any of the following activities in a coal mine, at least 90 days before beginning any of those activities, namely,**

(a) the development, construction or alteration of a portion of a coal mine;

(b) **the introduction or alteration of mining methods or technology;**

(c) **the use of new methods of construction, operation or installation of equipment;**

(d) the introduction or alteration of any main fan, booster fan, machine, equipment or instrument;

(e) the introduction or alteration of transportation equipment or a transportation system for employees or materials;

(f) the construction underground of a bulkhead, dam or permanent stopping;

(g) the construction of a tailing dam, a water storage reservoir or a structure above ground for the storage of explosives;

(h) the introduction of any electrical equipment or electrical system that has not previously been approved, or the alteration of any electrical equipment or electrical system; or

(i) the bulk transportation of explosives.

(2) **The notice referred to in subsection (1) shall contain pertinent detailed drawings, plans and specifications.**

(3) Where the drawings, plans and specifications referred to in subsection (2) are in respect of a submarine area, they shall indicate

(a) the mining method to be used;

(b) the area from which coal is to be extracted;

(c) sections of the strata from the sea bottom to the workings;

(d) the location of any faults or other geological disturbances; and

(e) the calculated strain on the sea bottom.

(4) For the purposes of subsection (3), a submarine area includes an area below any body of water or material that may flow.

The requirement for the submission of new procedures, e.t.c., to the Coal Mine Safety Commission (CMSC) is new to legislation governing CBDC operations, appearing for the first time in this revision. The introduction of a rock bolting trial falls into the category of "the introduction or alteration of mining methods or technology" and "the use of new methods of construction, operation or installation of equipment", highlighted above. In this case, CBDC are required to conform to Paragraph 5, as follows:

5. Where the employer submits plans or procedures in respect of a coal mine to the Coal Mining Safety Commission for approval, the employer shall

(a) without delay after the submission, notify the safety and health committee or the safety and health representative in writing of the submission; and

(b) keep a copy of the approved plans or procedures readily available at the coal mine for examination by the employees.

In addition to the statutory notification of both the CMSC and the safety and health representatives, Paragraph 35 of the regulations states requires that safe operating procedures be developed and implemented for each employee occupation as follows (emphasis by JWA):

35. (1) **The mine manager shall, in respect of each underground portion of a coal mine,**

(a) develop for employees procedures for the safe entry to, exit from and occupancy of that portion;

(b) develop emergency procedures that include

i) an emergency evacuation plan

ii) a description of the procedures to be followed

iii) the location of the emergency equipment provided by the employer

iv) an up to date plan of the coal mine; and

(c) **develop and implement safe operating procedures for each employee occupation.**

(2) **The employer shall keep a copy of the procedures referred to in subsection (1) readily available for examination by employees at the coal mine in respect of which the procedures apply.**

(3) **The employer shall instruct and train all employees in the procedures referred to in paragraphs (1)(a) and (b).**

The District Manager at the Sydney office of Labour Canada expressed the opinion that although rock bolting might prove to be a viable alternative to the present steel support systems, its' introduction should be undertaken in a controlled and monitored fashion.

3.2 Discussion

The new Occupational Health and Safety regulations now governing CBDC operations are very explicit in their coverage of new mining technologies but not in their treatment of strata control. Rock bolting as a supplementary support is likely to be treated as a new technology, and therefore CBDC are required to submit details to the Coal Mine Safety Commission 90 days before the trials commence. Workers representatives must also be informed. In addition, safe working procedures must be developed, training given and procedures made available at the workplace.

These requirements dictate an organized preparation for and implementation of rockbolting as a support method. Such a method is described in the following chapters.

4.0 IDENTIFICATION OF TRIAL SITES

4.1 Selection Process

4.1.1 Initial Consultation

Discussions with CBDC staff were held to identify the type of location at each mine that satisfied the trial requirements. The representativeness and accessibility of the sites, existing mining methods and the mine infrastructure required to complement the rock bolting trial were considered during these discussions.

4.1.2 Preliminary Site Selection

On the basis of the consultation with CBDC, preliminary site selections were made for each of the three mines. Meetings were arranged with the colliery engineer at each mine. Potential rock bolting trial sites were discussed, and the locations of potential trial sites solicited. Underground visits were then made to examine the proposed sites.

4.1.3 Site Visits/Final Selection

Site visits were made to the preliminary sites identified in conjunction with the colliery engineers. Each of the preliminary sites were examined and the suitability of the site or area for final selection judged on the basis of geotechnical conditions and existing infrastructure.

4.2 Lingan Colliery

At Lingan Colliery, Mr. Zygmunt Fuginski described two potential locations for the rock bolting trial; the 13 East Bottom Level Crosscuts between the deeps and the proposed driveage to connect the materials haulages on 2 Deep and 4 Deep. The 13 East Bottom Level crosscuts will be driven with an arched profile, but the Deeps Connector Drive may be a squarework driveage. However, the future development plans of Lingan Colliery are in a state of flux, and we are advised that either or both of these driveage plans may change in the near future.

The Deeps connector is not a production driveage and so the pressure to complete it may be less. In addition square profile roadways can generally be bolted with less difficulty than arched roadways. However, the 13 East Bottom Level offers certain advantages over the Deeps connector; it is the only arched roadway site of those initially proposed, and as it is proposed to serve two retreat faces as a dual life gateroad, CBDC are already considering extra reinforcement in the form of rock bolts.

4.3 Phalen Colliery

At Phalen Colliery, Mr. John Reid described several potential sites in both the gateroad driveages for the retreat walls and in the development headings. However, he expressed doubt that the requirement of installing the bolts immediately the roof was exposed (ie. in front of the driveage machine) could be met,

due to the difficulty of manoeuvre and operating rock bolting drills in front of the cutter. However, he mentioned that in the near future a modified VME Roadheader would be acquired and located in the 5 East Top Level. This machine, which is currently being assembled on the surface at the colliery, has powered supports behind the cutter. The presence of these supports will serve two functions; to fully support the newly exposed roof prior to the installation of standing supports, and to allow the standing supports to be installed behind the face independently of the cutting operation.

The introduction of this mining machine represents a good opportunity for a rock bolting trial at Phalen Colliery. The separation of the cutting and support functions will more easily allow the introduction of bolting technology by eliminating interference with the productivity of the heading, which was a requirement stipulated by CBDC at the initial contract meetings.

4.4 Prince Colliery

Conversations with Mr. Cyril LeBlanc at Prince Colliery highlighted the longwall development driveages as potential sites for the rock bolting trial. However, he had reservations about using a production gateroad because of the possibility of the trials delaying the critical development schedule. As an alternative, he suggested that a gateroad be extended past the proposed faceline for a distance of 30 to 50 metres for the purpose of the rock bolt trial. It is anticipated that the stub would be accessible for monitoring for a period of between four and nine months, depending on the schedule of operations, during which time the support system variations would be monitored. The exact location of the stub could not be determined in view of the lead time required by safety and health regulations and the uncertainty of development progress.

4.5 Documentation of Site Conditions

4.5.1 Mining Environment

The mining environment established or anticipated at each site was determined during the site visits. A summary of the features of each site are given in Table 4.1. In addition to the details of mining method, support type and available infrastructure, geotechnical parameters applicable to each site were obtained from previous work in the coalfield. These parameters included geomechanics classifications typical of the trial site areas and in-situ stress determinations.

4.5.2 Geology

Details of proximate geology at each site are difficult to determine in advance due to the variable nature of the strata in the coalfield, although extrapolation from existing mapping is possible to some extent. The approach adopted has been to refer to previous work reported by CANMET (1989) in which a number of sites at all three collieries were investigated as part of a study of rock mass classification systems. The various types of roof strata identified in the coalfield had previously been identified by CBDC geological staff (Figure 4.1). The CANMET study sites were identified in terms of the roof strata type and classification

ratings developed from field mapping. By examining the roof types and classification ratings developed for each colliery, anticipated roof conditions at each site were obtained.

4.5.3 In-Situ Stress

Experience in Australia and the United Kingdom has shown that in-situ stresses are important factors in the behaviour of rock bolted roadways. Roadway roof conditions caused by the effects of lateral stresses are illustrated in Figure 4.2. In situ stress measurements in the Sydney Coalfield have been undertaken using overcoring and hydrofracturing techniques. The results of these measurements were compiled by CANMET (1989), and are presented in Table 4.2.

4.5.3.1 Lingan Colliery

In-situ stress measurements at Lingan Colliery determined that the horizontal stresses in the E-W direction were approximately twice the N-S and vertical stresses. The 13 East Bottom Level will be driven a few degrees south of east in virgin ground, i.e. almost parallel to the maximum stress. If the principal stress direction remains the same, then, referring to Figure 4.2, the 13 East Face side of the level will be expected to suffer no ill effects. However, if the stress direction varies to the north, poor roadway conditions can be expected. As the stability of this side will determine to a great extent the ease of working of the 13 East Face, it might be prudent to ensure that the bolting pattern provides additional support on this side of the gateroad.

4.5.3.2 Phalen Colliery

In-situ stress measurements have not been carried out at Phalen Colliery, although they are scheduled to be performed in mid-1990. Because the Phalen workings lie immediately below the Lingan workings, it is assumed that the principal stress directions will be similar. At Lingan Colliery the horizontal stresses in the E-W direction were approximately twice the N-S and vertical stresses. At Phalen colliery the 5 East Top Level will be driven a few degrees south of east in virgin ground i.e. almost parallel to the maximum stress. If the principal stress direction remains the same, then, referring to Figure 4.2, the pillar sides of the levels will be expected to suffer no ill effects. However, if the stress direction varies to the north, poor roadway conditions can be expected. Once again, the bolting pattern must take this into account.

4.5.3.3 Prince Colliery

In-situ stress measurements (overcoring method) at Prince Colliery determined that the horizontal stresses were approximately twice the vertical stress. Stresses determined from hydrofracturing indicated a slightly elevated E-W horizontal stress, which conforms to measurements made at both Lingan Colliery and the Donkin Morien project. Panel gateroads on the west side of the Prince Colliery are driven from east to west in a WNW direction. If the principal stress direction remains as described then, referring to Figure 4.2, the north walls of the gateroads will be expected to suffer from poor conditions. The bolting pattern must take this into account by providing additional support on the stressed side of the gate.

4.5.4 Geotechnical Conditions

Geotechnical conditions can be determined by using any one of a number of geotechnical classification systems. A recent study (CANMET, 1990) compared a number of geotechnical classification systems for applicability in coalfield conditions.

The CANMET work generally concluded that the RMR system provided better estimations of support pressure when compared to the observed support deflection. This was attributed to the derivation of the equations from coal mine case histories. In nearly every case, however, support pressures determined from RMR ratings were greater than those obtained from beam deflection measurements. This was attributed to plastic deformation prior to support installation, the effect of wooden blocking in absorbing energy released, and inconsistencies in support installation. Similar experience has been obtained in the U.K. during recent rock-bolt design studies.

4.5.4.1 Lingan Colliery

Geomechanical mapping at Lingan Colliery revealed four types of roof, types C,F,G, and H as illustrated in Figure 4.1. The RMR ratings established for these sites ranged from 44 to 70. Ratings developed for the east side of the Colliery, away from sandstone channels in the roof, were considered to be more representative of the conditions likely to prevail in the 13 East Bottom Level. The typical roof types on the east side were C, F and H, with an average RMR value of 45. Estimated support pressures, determined from both the deformation of the arches and the RMR values, were reported to be in the range 19 kPa - 66 kPa, with mean values of 36 kPa for those determined from deformation measurements and 63 kPa for those determined from RMR values.

4.5.4.2 Phalen Colliery

Geomechanical mapping at Phalen Colliery revealed seven types of roof, types B,C,D,E,F,G, and H as illustrated in Figure 4.1. The RMR ratings established for these sites ranged from 41 to 91. These sites were on the west side of the colliery. The predominant roof type on the east side of the Colliery is expected to be type E. This roof type gave the lowest average RMR ratings (average 48) of the roof types studied by CANMET. The selection of the weakest roof material for design purposes will provide a greater margin of safety for the trial. Estimated support pressures, determined from both the deformation of the arches and the RMR values, were reported to be in the range 0 kPa - 57 kPa, with mean values of 2.2 for those determined from deformation measurements and 48.1 for those determined from RMR values.

4.5.4.3 Prince Colliery

Geomechanical mapping at Prince Colliery revealed six types of roof, types A,B,C,D,E and F, as illustrated in Figure 4.1. The RMR ratings established for these sites ranged from 35 to 81. Ratings developed for the predominant rock types in the west side of the colliery, B and C, range from 44 to 57, with an average of 52. Estimated support pressures, determined from both the deformation of the arches and the RMR values, were reported to be in the range 0 kPa - 434 kPa, with mean values of 102 kPa for those determined from deformation measurements and 54.0 for those determined from RMR values.

4.6 Discussion

The sites identified above were selected as a result of discussions with a number of CBDC representatives. They represent locations that are of most concern in terms of strata control, or which are least critical in terms of production scheduling.

The 13 East Bottom Level at Lingan Colliery is scheduled to be a dual life, arched gateroad. Several factors are present which make this a poor candidate for a rock bolting trial. These are:

- The shape of the roadway - cutting of the arch destroys the integrity of the roof beam.
- Horizontal stresses required to maintain a rock bolted beam will reduce to almost nothing when the first face line passes due to relaxation allowed by formation of the waste.
- The effect of two sets of face front abutments on the supports will have a severe effect on control of the gateroad.

Whilst there are concerns for the control of this gateroad in view of its importance to the success of retreat working at Lingan Colliery and the viability of the colliery, it is not a good site for a rock bolting trial leading to sole support on rock bolts. Rock bolts may have a beneficial effect on strata control, however, and the example calculation of a rock bolt pattern given in Appendix 1 shows how such a pattern can be designed.

The proposed Deeps Connector Drive at Lingan Colliery would provide a better site for a rock bolt trial at this colliery. Not only is it scheduled to be driven with a rectangular profile, but it may also be less critical in terms of production schedules. Success at this location may also show that rectangular deeps development using rock bolts and light steel support may be an option in the future. Elimination of the arch profile above the seam by driving rectangular roadways in and below the seam may also play a part in reducing outburst hazards.

The preferred site at Phalen Colliery was the 5 East Top Level, in conjunction with the new VME Roadheader. Whilst some concern over the implementation of two new technologies in the same heading has been expressed (Stewart, 1990), and and also concern over the damaging effects of the powered supports on the roof strata (Haslett, 1990), implementation of rock bolting trials at this location is still preferred for the following reasons:

- Discussions at Phalen Colliery raised doubts about the safety of placing rock bolts in front of conventional driving machines. Early, active support is a fundamental principal of effective rock bolting.
- The new VME Miner will support the roof between cutting and installation, reducing deterioration and allowing rock bolts to be installed along with other support behind the miner.
- The mean load densities applied by the powered supports behind the cutting head can be regulated to ensure that sufficient support is provided without excessive damage to the strata.

- Installation of supports behind the miner will reduce cutting downtime, and will be more acceptable to CBDC engineers than reductions in productivity caused by waiting for rock bolts to be installed.

- Implementation of new driveage technology may make the concept of associated supplementary rock bolt support easier to introduce.

At Prince Colliery, the idea of reducing interruptions to development by driving a special rock bolted heading has merits from an operational point of view, however it will delay full implementation of rock bolting as a support method. The review of mining and geotechnical conditions shows that Prince Colliery has the best conditions for the trial, with the exception of compressed air volume and quantity. However, hand held hydraulic rotary bolting drills powered from the Dosco Dintheader installing bolts over the first 100 m of the gateroad (ie. the furthest from the face start line) would allow the long term performance of the bolts to be monitored and would not affect the performance of most of the coal face should abutment problems develop. If the system performs well, a decision to continue the trial along the whole of the gateroad could still be made. Alternatively, the last 100m before the face start line could be used for the bolt trial. Whilst limited in length, this would provide for earlier results and implementation in other driveages.

5.0 DETAILED DESIGN OF A ROCK BOLTING TRIAL

5.1 Design Strategy

The detailed design of a trial bolting system involves a number of complex, inter-related tasks. The most recent example of the implementation of a comprehensive rock bolting strategy in coal mines for which published information is available was undertaken in the United Kingdom. An ordered implementation strategy was directed from British Coals' Headquarters Technical Services Department at Bretby after a comprehensive review of overseas experience and available resources within the coalfields. This programme was described by Buchanan et. al. (1989). The principal features of the programme implemented by British Coal were firstly the considerable effort expended in ensuring that the most up-to-date theories, technologies and experiences from around the world were considered, and secondly, the resulting thorough, stepwise approach to the implementation of rock bolting. At an early stage in the study, rock bolting professionals from industry, government and coal operators in Australia, the U.S.A. and Europe were consulted, and Australian and U.S. researchers were invited to conduct trials in the U.K. The overall outcome of the consultations and trials was the development of a nationwide, centrally directed roof bolting strategy. This strategy has resulted in the successful introduction of rock bolting within the U.K. mining industry.

Despite the geographical distance between the coalfields of the U.K and Cape Breton Island, there are many similarities both in the mining environment and the reasons for enhanced interest in rock bolting as a means of support. At both locations the predominant method of extraction is by longwalls served by single entries on either side. In Cape Breton, as in the U.K., retreat faces with single or dual use gateroads are becoming the extraction method of choice for reasons of efficiency and productivity. The use of rock bolts as a supplementary support to augment the steel arches and to allow increased arch spacing, or ultimately as the sole support method under suitable geological conditions, offers further benefits to mine economics and productivity, as indicated by the U.K. experience (Buchanan, et al. 1989). The strategy adopted for the detailed design of a trial bolting system for the Sydney Coalfield is thus based heavily on the successful strategy employed in the U.K., and as such includes the most recent practical experience in the implementation of such a programme in the coal mining environment.

In order to complete a comprehensive and detailed design, several factors must be considered;

- methods of assessing the support requirements at trial sites must be established, along with the bolt pattern design procedures to be used to ensure that these support requirements are met,
- equipment required to install the rock bolts must be selected for safety and suitability in the individual site environments.
- monitoring and analysis programmes established to determine the effectiveness of the rock bolts as a support.

- training programmes required to ensure efficient and effective implementation of the trial, and

Each of these factors is considered in turn below.

5.2 Support Requirement Assessment and Pattern Design

5.2.1 Assessment of Support Requirements

The assessment of support requirements at individual locations can be determined analytically or empirically. Analytical methods, such as those published by Wilson (1983) or Hoek and Brown (1980), relate hydrostatic in-situ stress levels, rock properties parameters and simple opening geometries to the development of a yield zone around the opening, the development of convergence and generated support pressures. Empirical methods (Bieniawski, 1979, 1984; Barton, et. al. 1974, 1977) are based on rock mass classification systems and relate a scalar rock mass properties rating to anticipated support pressures, unsupported distance and stand-up time in a relationship based on theory or back analysis of previous construction.

5.2.1.1 Analytical Methods

Both analytical methods mentioned above depend heavily on assumptions made regarding the rock failure criterion to determine the position of the yield-elastic boundary. Both rely to some extent on elastic or elasto-plastic theory for the calculation of convergence, although Wilson's method is also based on the observed behaviour of broken rock masses.

The failure criterion used by Wilson can be expressed as;

$$\sigma_1 = \sigma_0 + k \cdot \sigma_3 \qquad (5\text{-}1)$$

where; σ_1 = stress at failure (MPa),
σ_0 = unconfined compressive strength (MPa),
σ_3 = confining pressure (MPa), and
k = triaxial stress factor, given as;

$$\frac{(1 + \sin\Phi)}{(1 - \sin\Phi)} \qquad (5\text{-}2)$$

where Φ is the angle of internal friction of the rock (degrees). This failure criterion depicts the triaxial failure envelope as a straight line. Typical triaxial stress factors for coal measures rocks are well established.

The failure criterion used as a basis for the Hoek-Brown analysis was developed from the analysis of triaxial tests of a large number of rock types;

$$\sigma_1 = \sigma_3 + (m \cdot \sigma_0 \cdot \sigma_3 + s \cdot \sigma_0^2)^{0.5} \qquad (5.3)$$

where σ_1 , σ_0 and σ_3 are as defined in (5-1) and "m" and "s" are material constants related to the fundamental rock properties and the extent to which it has been broken prior to being subjected to the stresses σ_1 and σ_3 .

5.2.1.2 Empirical Methods

Empirical methods of assessing support requirements, rock mass classification systems and their derivative theories, were extensively reviewed by CANMET (1989) for application to the Sydney Coalfield. Aston and Gallant (1988) also reviewed empirical methods in an effort to derive a bolting pattern for cross-cuts at Prince Mine.

The two most commonly used classification systems used to determine support pressures are the NGI system (Q rating) and the CSIR system (RMR rating). Both CANMET (1989) and Aston and Gallant (1988) studied these systems, and these and other references can be used to obtain further details. Outlines of both systems are given below.

The NGI classification derives a numerical rating, Q, based on the assessment of six parameters, as follows;

- RQD; rock quality designation
- J_n ; joint set number
- J_r ; joint roughness number
- J_a ; joint alteration number
- J_w ; joint water reduction factor
- SRF; stress reduction factor

The rock mass rating, Q, is given by the following expression;

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \qquad (5\text{-}4)$$

In conjunction with the Opening Equivalent Dimension, D_e, determined as the excavation span divided by the Excavation Support Ratio (ESR - roughly analogous to the inverse of the factor of safety), approximate support pressures and corresponding support requirements can be found by reference to tables compiled from back-analysis of a large number of excavations.

The CSIR system produces a Rock Mass Rating (RMR) by summation of scalar parameters reflecting;

- uniaxial compressive strength,
- RQD,
- discontinuity spacing,

- discontinuity condition,
- groundwater conditions, and
- orientation of the discontinuities.

The rock mass is then classified into one of five ranges from very good to very poor depending on the value of RMR. Theoretical support loadings are then calculated based on the weight of rock in the loosened zone above the opening, the height of which, h, is given as (Unal, 1983);

$$h = \frac{100 - RMR}{100} \times S \tag{5-5}$$

where S is the span of the opening.

In the U.K., modifications have been made to the CSIR method, giving the smooth planar joints often found in coal measures mudstones a rating of 15, and giving even the driest excavations a rating of 10 (damp).

5.2.2 Rock Bolt Pattern Design

Many authors have tried to rationalize the effect of rock bolts on rock masses to derive pattern design methods. However, a definitive theory has yet to be found, and design methods remain based on empirical methods or simple analytical methods based on simple reinforcement mechanisms. The exact nature of the rock bolt/rock interaction in-situ remains unknown.

Aston and Gallant (1988) reviewed several methods of pattern design. Analytical methods evaluated included those by Panek (1964) and Bieniawski (1984). Empirical methods reviewed were those devised by the U.S. Army Corps of Engineers (1980), the NGI System (Barton et al, 1974) and the French design guidelines (Raffoux and Dejean). The review of pattern design methods undertaken for the U.K. trials included all of the above methods, and an alternative analytical method described by Daws (1987). These are described below.

5.2.2.1 Analytical Methods

Panek (1962) proposed a method of rock bolt pattern design based on the friction induced by rock bolts between laminae of rock in a composite beam forming the roof of the opening. According to elastic theory, bending of a clamped roof beam decreases as the beam thickness increases. The maximum bending strain in a clamped beam is given by;

$$\xi_{max} = \frac{\rho . S^2}{2 . E . t} \tag{5-6}$$

where ξ_{max} = maximum bending strain
ρ = unit weight of rock (MN/m^3),
S = span of the opening (m),

E = elastic modulus of the rock (MN/m^2), and

t = strata thickness (m).

Rock bolts cause laminated strata to be bound together to form a composite beam of increased thickness, and therefore beam bending and elastic strain on the abutments is reduced, contributing to opening stability. Panek (1962) produced a nomogram for reinforcement design using this effect, however his work assumes elastic beam behaviour and ignores horizontal stresses which affect the behaviour of the beam.

Bieniawski's analytical method for bolt pattern design is based on the CSIR classification system and the calculation of rock load height from the RMR value. The method assumes a lower limit of lateral stress necessary to maintain a roof arch of 1.15 MPa. Lateral stresses are expected to be higher than this, as indicated by in-situ stress measurement results in the coalfield (Table 4.2).

Using this method, the bolt length, L, is determined from;

$$\frac{h}{2} \qquad (5\text{-}7)$$

or from;

$$S\,\frac{2}{3} \qquad (5\text{-}8)$$

whichever is the greater, where;

S = opening span in metres, and

h = height of broken strata (m) calculated from the RMR value as in Equation (5-5.)

The bolt spacing, S_b , in metres, is determined from the following;

$$S_b = \left(\frac{C_b}{F \cdot \rho \cdot h}\right)^{0.5} \qquad (5\text{-}9)$$

where; C_b = bolt capacity in MN (smaller of steel yield load or anchor capacity)

F = Factor of Safety

ρ = rock density (MN/m^3), and

h = height of broken strata in metres (Equation 5-5)

The bolt capacity is an important parameter in the equation and some investigation is required to determine the appropriate value. Bolts will only achieve their yield load if the grouted annulus retains the bolt in the hole to this load. In-situ test must be conducted on relatively short anchorage lengths to determine anchorage capacities for different strata types. If it is found that the anchorage capacity over the length of the bolt is large enough to achieve loads greater than the yield load, then use of the yield load in the equation ensures that the system operates in the linear-elastic portion of the steel stress-strain properties, and prevents plastic deformation over extended periods of time.

Daws' design method equates the thrust generated by the steel arch to the increase in strength generated in the rock mass by the confining effect of the rock bolts. By making an analogy between the increase in rock strength due to confinement in a triaxial test with the increase in rock strength caused by the confinement offered by rock bolts, and using the Mohr Failure criteria, Daws produced the following design equation for rock bolts;

$$\left(P_a \,.\, S_b\right) + \left(k \,.\, C_b \,.\, S_b\right) - \left(k \,.\, C_b \,.\, L\right) = 0 \qquad (5\text{-}10)$$

where; P_a = active support pressure (MN/m^2),

S_b = bolt spacing (metres - square grid assumed),

C_b = yield load of the bolt (MN),

L = bolt length (metres), and

k = triaxial stress factor (Equation (5-2)).

In calculating the bolt pattern using Daws' method certain input values must be known or assumed to solve the quadratic equation. Adopting a value for k of 3, a bolt yield load of 0.2 MN and a bolt spacing of 1 m, the equation can be solved for L by substituting a value for Pa, the active pressure. This can be either that offered by the standing support, or that estimated from the RMR values. Input parameters can be varied until an optimum design is reached.

5.2.2.2 Empirical Methods

The U.S. Army Corps of Engineers empirical design guidelines (U.S. Army Corps of Engineers, 1980) specify minimum design parameters for rock bolts above and below the springline of the opening. Rock bolting below the springline is not practised in coal mine gateroads, so the requirements can be summarized as;

- bolt length: greatest of;
  - 2 times spacing,
  - 3 times thickness of unstable blocks, or
  - 0.5 times span (for openings m).
- maximum bolt spacing: least of;
  - 0.5 times bolt length,
  - 1.5 times width of unstable blocks, or
  - 2.0 m (else lagging is difficult to secure).
- minimum spacing: 0.9 to 1.2 metres.
- minimum average confining pressure: greatest of;

- vertical rock load of 0.2 times span, or
- 40 kN/m^2.

The empirical guidelines put forward as part of the NGI system are in the form of a table. The recommended supports are based on documented case studies from the civil construction and hard rock mining industries, but are very general and do not indicate bolt lengths. In many cases, shotcrete is recommended as a supplement to the rock bolt reinforcement. Taking as an example the anticipated conditions in the 13 East Bottom Level at Lingan Colliery, i.e. a Q value of between 3 and 6, an average value of RQD/J_n of 11, an average value of J_r/J_a of 0.34 and a Span/ESR value of between 1 and 2 (CANMET,1990), the NGI support category would be 18 or 21, requiring untensioned grouted bolts at 1 to 1.5 m spacing and chain link mesh lagging or 2.5 to 5 cm shotcrete support, respectively.

The French bolting guidelines illustrated in Figure 5.1 give the following design parameters for typical conditions in the Cape Breton Coalfield;

- medium density, (0.75 to 1.25 bolts per square metre)
- heavy lagging (heavy gauge wire mesh or weld mesh)
- medium bolt length, or 0.3 to 0.5 times the span. In this case the span is 6 metres, requiring a bolt length of 2 - 3 m.

The French guidelines also give some general advice with regard to the best locations for the use of rock bolts as a support method. Rock bolting should not be used if;

- more than 20% of the first 5 m. of roof strata is coal,
- bed separation in the first 2 m. of strata is liable to exceed 5 cm.,
- the maximum thickness of differing strata bands is less than 20 cm, or
- the roadway is liable to be subjected to the effects of interaction from other workings.

5.3 Selection of Equipment

Rock bolting equipment can be broken down into the following categories;

- drills, rod, and bits
- bolts, plates, nuts, washers
- resins or resin substitutes
- additional roof support components

Each of these items must be selected for safety and suitability in each trial environment.

5.3.1 Drills, Rods and Bits

The selection of appropriate drills, rods and bits is a major factor in the outcome of the trial. The installation time for rock bolts is largely governed by the speed at which an appropriate hole can be drilled; this depends on the capabilities of the drill, the condition of the bit and the efficiency of rod changes, and the skill of the workman.

Unfortunately, mechanised drilling technologies appropriate for use in the restricted space of single entry coal mine development headings have not kept pace with the reintroduction of rock bolts. In France and Germany, bolted heading sizes are typically much larger than those in the U.K. and Cape Breton, and in the U.S.A and Australia, mechanised room and pillar multiple entries allow the cyclic use of single purpose machines in headings.

Telescopic rock bolting booms are available for retro-fitting to roadheaders. Torque Tension of the U.K. supply such a device. Wheel or crawler mounted drilling machines are also available - Mindev and Hausherr are two examples. However, these are considered too bulky and costly for acquisition for a trial. The alternative to this type of equipment is to use hand held drilling equipment.

A number of hand-held drilling machines have been tried with varying success in the U.K. and Australia. Traditional rotary percussive stopers such as the Atlas Copco Falcon are not appropriate because they generally provide poor torquing capabilities during tensioning of the bolts. Recent trials with the Cram Wombat at Phalen Colliery were reportedly successful, and it is likely that similar machines could be used successfully throughout the coalfield. However, these machines demand high volumes of compressed air at a constant pressure for optimum drilling rates. Typical compressed air requirements are 3 - 3.6 m^3/min at 0.5 - 0.7 MPa. Typical drilling rates are 1 m/min for a 32 mm. diameter hole (twice that speed for a 27 mm. diameter hole), provided that good quality drill bits and water flushing are used.

Experience at Kellingley and other collieries in the U.K. has been that the rotary compressed air type of drill requires considerable maintenance, despite the many measures taken to ensure clean and lubricated compressed air. In order to maintain economic drilling and bolt installation rates two spare machines are kept for every one in operation. One spare is kept at a meeting station close to the face, whilst the other is usually on surface undergoing repairs. Similar experience was recorded in Australia. In the absence of other drilling alternatives, this approach is recommended for the Sydney Coalfield trials if advance rates are critical.

Hydraulic rock bolting drills are generally heavier than their compressed air counterparts, but recent advances in the safety of composite materials in coal mines contribute to keeping the weight to manageable proportions. In most cases, the hydraulic drills can be powered from spare circuits on roadheading machines, as long as pressure reducers and flow controllers are installed where necessary to ensure that the manufacturers recommended fluid supply criteria are met. The use of hydraulic drill machines operated from the faceline hydraulic system has not always been so successful due to the high back pressures on the return lines. Drilling rates are comparable with the compressed air machines.

Drill machine manufacturers will recommend the most appropriate size of machine (air leg extension capacity) if given details of the bolting plan at the time of ordering. In addition to the purchase of drills,

ancillary equipment such as oilers, filters and spares should be obtained to cover the duration of the trial. Manufacturers recommended compressed air or hydraulic fluid volumes and pressures should be maintained at all times during the trial. Prior to the start of the trial, colliery engineers should ensure that air supplies and power packs are in order in the trial headings.

For installation of rib-side bolts, conventional coal borers with scroll rods and coal bits of the correct diameter as presently used in the Sydney Coalfield should be made available in the headings.

The selection of the drill bit is an important factor in the trial. Bit diameter and sharpness affect hole diameters, the efficiency of the resin bonding and drilling rates. Drill bits must be maintained in good condition by either sharpening or replacement. If bits are to be sharpened, they must be examined and discarded if the wear is excessive, i.e. if they will not drill a hole of the required diameter. If bits are to be discarded, sufficient supplies of bits must be available at the worksite to maintain drilling rates.

The preferred type of bit for drilling in the roof is a two wing bit with thick, "tough" grade carbide inserts and a negative rake. Coal bits are usually neutral or positive rake and do not give good results in stone. Bits should be equipped with a central hole for water flushing to be used. Bit suppliers such as Kennametal, Celtite, Boart, Padley and Venables e.t.c., should be contacted for specific recommendations, costs and delivery times.

Two or three pass drilling will be required to drill holes of sufficient length for bolt installation, depending on the extension of the drill leg, the drill dead length, and the required hole length. In practise, it is more efficient to use drill rods of several different lengths rather than snap-on or screw connected rods. A short, 1 m long starter rod ensures effective collaring and improved safety. Each rod is equipped with a suitable bit (see above), maintained in good condition.

To allow for fitting of the nut, the drilled hole length, i.e. the length of the finishing rod, should be 50 mm shorter than the bolt length. Using water flushing there is no need to use scroll rods to clear cuttings from vertical or near vertical holes in the roof, and smaller, lighter, 22 mm hexagonal steels can be used. Spare rods of each length and drill bits should be kept in the heading in reserve.

5.3.2 Bolts, Plates, Nuts and Washers

5.3.2.1 Selection of Bolt Type

Rock bolts can be generally be classified as;

- untensioned grouted dowels,
- mechanical, point anchored,
- grouted, point anchored,
- friction stabilizers, or
- tensioned grouted dowels

Each of these types has found acceptance in mining at some time or another, but not all of them are appropriate for use in coal mine conditions. Detailed descriptions of each of these types and the many varieties within each type can be found elsewhere (Hoek and Brown, 1980, Stillborg, 1986).

Untensioned grouted dowels (Figure 5.2) are popular reinforcement in metal mines; strata movement is elastic, occurring soon after the opening is formed and bolts are used to secure blocks that loosen over an extended period of time. In coal mines, where significant plastic deformation occurs, untensioned dowels do not provide immediate support. Some deformation has to take place to generate load, and hence resistance to movement. All relevant experience in coal mines shows that early, effective support is essential to ensure control of the strata surrounding the opening. For this reason alone, untensioned bolts have been rejected as the preferred method of providing roof support in coal mine roadways, although wooden or fibreglass untensioned dowels are essential to stabilize coal ribs.

Point anchored bolts, either grouted or mechanical, can be pre-tensioned to provide immediate support, but these types also have their disadvantages. Quality control and long term performance of point anchored bolts are difficult to standardize. With grouted point anchor bolts, the location of the anchor point can vary significantly due to movement of the resin capsule or mixed resin flowing out of the hole under gravity. Mechanical point anchors must be carefully matched to the strata conditions so that the pressure exerted by the bearing surfaces does not cause penetration of the borehole sides and hence reduction in support capacity. With both types, significant load fall-off can occur soon after installation, reducing the reinforcement effect.

Friction stabilizers, such as "Swellex" (Atlas Copco) or "Split Set" (Ingersoll Rand) are a relatively new concept offering easy installation and low cost. However, their tubular construction reduces their tensile strength, and they are more susceptible to shear and strength reduction due to corrosion. In some cases, ability to accept shear deformation without failure may be an advantage, but in a coal mine roadway, where significant inter-layer shear can destroy strata control, they are generally not applicable. The Swellex bolt has a considerable advantage over the "Split Set" in that installation does not require a drill and is relatively quick and easy. It is also widely used at present in the Sydney Coalfield and therefore has a high level of acceptance.

Tensioned grouted bolts have been very successful in providing support in coal mines in Europe, Australia, and the U.S.A. The bolt is usually of conventional concrete reinforcing bar, with a thread cut or cold-rolled at one end to accept the nut. Their use with quick and slow set resins or resin substitutes gives a rapid and secure anchorage. Fast set resin at the end of the hole allows tensioning at the time of installation. The slow set resin then bonds the tensioned length to the rest of the hole. Complete filling of the hole also helps to prevent corrosion problems from developing.

Load bearing capacities of this type of bolt depends upon the diameter and the grade of steel used. Grade 400 rebar has a specified yield stress of 400 N/mm^2, giving yield loads of 20 tonnes and 12 tonnes for 25 mm and 20 mm bolts respectively, and failure loads of 30 tonnes and 18 tonnes respectively. In order to achieve these loads, the anchorage capacities of the resin grout in the rock must be able to withstand these forces. Testing of anchorage capacities is dealt with in a later section.

Tensioned grouted rock bolts were selected by British Coal for their centrally directed rock bolting trials in a number of coalfields. These trials were based on extensive collaboration with Australian, French, West German and U.S. operators, utilizing the best experience and techniques from each. The dimensions of the bolts used in the initial trials were 25 mm diameter in 32 mm to 35 mm diameter boreholes. In the Western Area it was intended from the outset to install 20 mm rebars in 27 mm diameter boreholes using 24 mm diameter resin capsules. However, because of the lower load bearing capacity of the smaller diameter bolts, these were only to be used if conditions allowed.

Following colliery trials, an Australian designed reinforcing bar is now used exclusively where rock bolts provide the sole means of roadway support in U.K. Collieries. Two grades are available, 460 and 700 N/mm^2 yield stress, giving typical yield loads of 17 and 27 tonnes respectively. They are installed in 27 mm boreholes, allowing for quicker drilling installation times. Initially imported, but now manufactured under license, these "AT" pattern bolts have a round core with a diameter of 22 mm. and low profile ribs forming a left hand helix giving a maximum diameter of 23 mm. The helix assists in retaining the resin in the hole during installation. An M24 thread is rolled onto one end of the bars to accept the nut, and the other end is cut at a 45 degree angle to facilitate penetration of the resin capsules.

5.3.2.2 Plates, Nuts and Washers

Plates, nuts and washers are essential accessories to rock bolts. The plate acts as a bearing surface for the nut and allows tensioning of the bolt. The nut is tightened onto the plate using the drilling machine to impart the bolt tension. Bolt plates should be 150 mm x 150 mm x 10 mm in size, with a circular hole bored or punched in the centre. The hole size should be no more than 30 mm in diameter. Domed bolt plates are acceptable.

Washers are usually not required, but if bolts are not installed perpendicular to the rock face, wedge washers or a hemispherical washer fitting into a machined bowl on the plate must be used to ensure efficient tensioning and to prevent the locking nut being forced into the plate. Hemispherical washers are usually made of cast metal and fit onto either a domed bolt plate or into a hemispherical depression in the plate.

Bolts should be equipped with torque nuts. These nuts have either a plastic insert in the top of the thread or crimped threads that prevent the nut from tightening until the resin has set. The specification for the South Yorkshire Area of the U.K. includes a metal disc in conjunction with the nylon insert. The metal disc prevents plastic shards from fouling the threads of the bolt and preventing correct installation.

5.3.3 Resins or Resin Substitutes

Considerable attention has been devoted in the U.K. to the strength characteristics of the resin grout in the annulus between the rock and the bolt. Flame retardant resin capsules produce a grout with a lower strength and stiffness than the straight polyester resins used outside the coal industry, and they are also subject to greater variability in specified setting times (Daws, 1990). These factors lead to difficulties during setting and premature failure of the grout bond under load. The industry in the U.K. has had some success in convincing the Inspectorate that hazards due to ineffective rock bolt installation brought about by flameproof resins are greater than the fire hazard associated with straight polyester resins. Resins should

be selected in consultation with the manufacturer. Factors affecting resin set times are ambient temperature, resin handling, and correct mixing times. Fast set resins may set very slowly in cold conditions - specified setting times are usually for 20 degrees centigrade - and if ambient temperatures differ significantly (+/- 5 degrees) from this at the trial site, a different resin may be required. Resin set times should be tested at each trial site, and fairly frequently afterwards to ensure manufacturers compliance with specifications.

Resins must be handled and stocked with care. Older stock must be used first, and existing stocks should be warehoused in warm and dry conditions. Cases should be handled carefully. Resin is the most expensive component of the system, and wastage due to capsule damage can add up to 25% to resin costs.

Capsule retainers - plastic cages on the capsules with arms that engage the borehole walls when inserted - are useful accessories to ensure that the resin stays at the top of the hole between insertion and installation of the bolt. A capsule launching tube is also of assistance in roadways where it is difficult to reach the back.

5.3.4 Additional Roof Support Components

Additional roof support components make up the lagging and are installed with the bolts to prevent "guttering" between rows of bolts. Lagging currently installed with steel supports comprises corrugated iron sheets secured with wooden wedges. The steel sheets are effective at preventing loose material from falling into the roadway, but obscure the roof from view and are impossible to bolt through. Alternate lagging materials more amenable to rock bolting are wire mesh or weldmesh, with weldmesh in panels or strips the easiest to handle.

Weldmesh is most convenient if ordered in strips 15 to 20 cm greater than the arch spacing, and in lengths equal to approximately half the width of the opening span plus 15 to 20 cm. Each sheet of mesh can then be overlapped and held in place with a row of rock bolts. Weldgrip, of Sheffield in the U.K., and British Steel Corporation, produce weldmesh strips to any specification, with wider strand spacings at any requested interval to indicate bolt hole locations. Alternatively, interlocking panel designs are available, some with cantilever hinges to keep the individual panels in contact with the roof. Similar manufacturing capabilities may be available in Canada, and should be explored in order to obtain regular supplies at optimum cost.

Metal straps are slightly more difficult to handle, as are wooden planks, but form an essential part of the rock bolting system if it is to progress to the stage of a sole support method. Straps for use in flat-topped roadways have a flattened "W" shaped profile, which provides additional stiffness. Predrilled holes define the rock bolting pattern, but may prevent small alterations in hole location which may be required due to roof conditions. This is especially important in arched roadways where the machine cut profile is irregular, but is not so much of a problem in flat-topped roadways. Metal straps or planks should be used on the ribs.

5.4 Installation Procedures

The establishment of the correct installation procedures for the rock bolts is essential if the bolts are to function efficiently. The installation procedures should be taught to both workers and management/supervisors, and the strictest supervision maintained until the installation standards are fully met. Periodic reviews should establish that installation standards are being maintained.

Installation parameters critical to the success of rock bolting are as follows;

- time of installation; rock bolts should be installed as close to the face of the heading as possible and immediately after the completion of the cutting cycle,
- hole drilling; the hole should be of the correct diameter and length, clean and smooth, and in the correct position with respect to the pattern design for the roadway,
- resin use; sufficient resin capsules should be inserted in the hole in the correct order. In addition, quality control over resin properties should be maintained,
- bolt installation; spin times and torquing instructions must be complied with to ensure effective mixing without over mixing, and the bolts torqued using the drill handle or other approved method.
- plates, straps, lagging; all accessory support equipment provided for in the pattern design must be installed correctly at the same time as the bolts.

The use of two or three drill steels does not usually reduce drilling rates by much if two man drill teams are used, and if the following procedure is employed to drill and install the rock bolts;

- With starter steel in bolter, drill hole A to length of rod,
- Swing bolter over to hole B position and drill hole B using same starter rod. As hole B being drilled, finisher steel is made available alongside bolter,
- After drilling to extent of starter steel in hole B change to finisher steel and finish hole B. Swing bolter back to hole A and finish hole A,
- As hole A is being finished charge hole B with resin capsules. Prepare bolts with plate and shear nut ready for installation,
- On completion of hole A change drill steel for spinning adaptor. Swing bolter back to hole B and install bolt.
- Whilst fast set resin capsule is curing, charge hole A with resin capsules.
- When fast set resin capsule is cured, torque bolt in hole B to load with bolter. Install bolt in hole A.

The above sequence is repeated for any two adjacent holes, and optimizes machine moves and drill down time due to steel changes.

Tensioning of the rock bolts is most conveniently done using the drills. All can be equipped with a chuck lock or similar device to allow the bolt retaining nut to be tightened using the arm of the drill as a lever. Bolt tensions applied in this way are of the order of 5 tonnes, but this depends on the effort expended by the drill operator in tightening the bolt. Slightly built workers will not be able to apply the same tension as heavier workers. In order to ensure that all bolts are installed with the same tension, pneumatic torque multiplier wrenches are used in some collieries in the U.K. to tighten the bolts after installation.

The procedures developed to ensure the correct installation should be documented and be made available to the workforce during the training programme.

5.5 Monitoring and Analysis

The monitoring and supervision of the demonstrations is of great importance. It is therefore considered essential that at least one CBDC Headquarters Engineer be made responsible for the trials as a whole, assisted by two or three experienced supervisors seconded to him for the duration of the trials. This team, properly trained, would coordinate equipment and materials supplies, training, demonstrations and monitoring. Monitoring of the trial should include operational/technical monitoring, and geotechnical monitoring, as described below.

5.5.1 Technical Monitoring

Technical monitoring of the trial will determine the effectiveness of the bolting system in terms of both cost and efficiency. Technical monitoring will also assist in the early identification of start-up problems due to unfamiliarity with equipment and methods. Such problems can then be resolved quickly and effectively before they endanger the overall trial. The monitoring will also provide valuable information for fine tuning the system. Parameters of importance in such a monitoring program include;

- rate of advance
- time lag between excavation and support
- time required to driveage operations - cutting, support, maintenance, advancing services, delays attributable to outside factors and compliance with design standards.

The efficiency of the bolting operation can then be determined.

Members of the rock bolting team should be at the sites during every working shift for the first few weeks of the trial to spot potential difficulties and correct them before they become a major problem. They should take note of details as described above, and can also monitor the geotechnical stations. Simple forms should be provided so that data collected can be compiled and analyzed more easily.

Such monitoring will ensure full compliance to design standards in the early stages of the "learning curve" and will provide detailed information to be used in conjunction with the geotechnical monitoring results to assess the success of the bolting pattern in controlling roadway closure.

5.5.2 Geotechnical Monitoring

Geotechnical monitoring and in-situ testing at the trial sites prior to the implementation of rock bolting for the purpose of assisting in pattern design has already been mentioned briefly. In addition to this work, the effects of the rock bolts on strata control in the roadway must also be monitored. Ultimately, these results will provide baseline data for empirical design guidelines specific to the colliery or coalfield as a whole.

Geotechnical monitoring sites must be installed in unbolted sections to provide comparison information. At least two fully instrumented sites should be installed in bolted and unbolted lengths of trial roadway to obtain representative data, and additional convergence monitoring stations should be installed to ensure the representativeness of the results. An appropriate geotechnical monitoring program consists of the following basic monitoring installations and procedures;

- Stage 1: Site Investigation
  - mapping, core drilling, detailed logging and sample testing to obtain geotechnical data for support assessment.
  - anchorage capacity testing.
- Stage 2: Investigation of Current Support
  - Determination of the rock/support interaction and strata deformation processes in conventionally supported sections of roadway prior to the implementation of the rock bolting trial by the use of the following instrumentation and procedures;
  - extensometers to measure differential strata movement in the roof and floor,
  - load cells installed in individual support elements to determine support loads generated by strata movement,
  - closure monitors to monitor changes in the dimensions of the opening.
  - repeated visual inspection and photography of the site and surrounding areas.
- Stage 3: Investigation of Rock Bolted Sections
  - Determination of the rock/support interaction and strata deformation processes in rock bolted sections of roadway using the instrumentation and procedures described above.
  - Additional geotechnical monitoring stations should be installed whenever bolting patterns are changed or arch spacing increased.

- Results of the investigations should be displayed adjacent to the instrumentation sites so that workmen can see for themselves the effects of the support method.

The following sections describe the instrumentation and methods specified above.

5.5.2.1 Mapping

Mapping of the trial roadways should be undertaken to determine the likely roof strata and conditions. Existing mapping in adjacent roadways can be extrapolated to the trial areas. Mapping should include lithology, joint frequency, joint orientation, joint properties, and observations on strata behaviour and water. This information can then be used in conjunction with data gathered from the core drilling to determine the Rock Mass Rating and to calculate the support pressure.

5.5.2.2 Core Drilling and Core Logging

Core drilling should be conducted to obtain samples of rock from the immediate roof to a depth of at least five metres wnere possible. Coring can be done using lightweight core barrels (for example Craelius TT56) mounted on a compressed air bolting drill. The core diameter should match the diameter required for sample testing, but larger core diameters will result in better recovery. If sufficient samples for testing are not obtained from one hole, additional holes should be drilled.

On recovery from the core barrel, the core should be laid in boxes and natural fractures marked and counted. Fracture frequencies and RQD should be determined before transportation to the surface. At the surface, a full geotechnical log should be prepared including test sample locations, orientation, nature and properties of discontinuities including a graphic representation, and a lithological log.

5.5.2.3 Rock Properties Testing

The principle rock properties of importance in the support assessment and determination of the rock bolt pattern design are the uniaxial compressive strength and the angle of internal friction. If the Hoek-Brown method of determining closure is to be used, material constants, m and s, must be found by analysis of test results.

The uniaxial compressive strength can be determined economically and quickly using the Point Load Index method. The advantage of using this method is that it can be done using smaller core pieces than the conventional direct determination of compressive strength, and leaves more sample for the triaxial testing required to determine the othearameters.

Triaxial testing must be undertaken to determine the angle of internal friction and the Hoek-Brown material constants. Single stage or multiple stage testing can be used. The single stage tests give more accurate readings but require more samples to define the failure envelope. Multiple stage tests determine the complete envelope, but underestimate it. The choice of test method should be made on the basis of the number of samples available, and sufficient tests should be done to ensure representative results (usually three to five).

5.5.2.4 Anchorage Testing

Anchorage testing is conducted with conventional rock bolt pullout equipment; a bearing plate, hollow jack and pump, pull bar and a dial gauge to measure extension. Pullout tests on properly installed bolts will probably result in the failure of the bolt by yielding at the thread, where the cross sectional area is smallest. If bolt failure occurs before this, it will be due to one of the following reasons;

- failure at the resin/rock interface as a result of poor hole cleaning, substandard resin performance or failure of the rock,
- failure at the resin/bolt interface due to substandard resin performance, or
- substandard steel performance.

Correct drilling and flushing procedures will result in correct hole conditions. Substandard resin performance may be due to incorrect handling, out-of-range ambient temperatures, or poor quality manufacturing. Investigation of the first two possibilities should be completed, and if necessary, changes made to the type of resin and handling and storage methods, prior to referring the matter back to the manufacturers.

If failure of the rock is indicated, pullout tests should be conducted using short encapsulation lengths to determine the strength of the resin/rock bond. The total resistance of the resin rock bond should then be used in the calculation of support pressure as described above.

5.5.2.5 Extensometers

A variety of proprietary extensometers are available for determining differential strata movement around mine openings. In addition to models supplied by specialist manufacturers, a number of simple methods amenable to fabrication in colliery workshops are available.

A basic extensometer consists of a wire anchored by wedges in a hole in the roof, floor or side. The wire extends through a plug, and changes in strata separation measured between a knot or block on the wire and the plug (Figure 5.3). Strata separation between the anchor location and the mouth of the hole can be measured to the nearest millimetre with a steel rule.

The Irad Gage Sonic Probe extensometer currently held by CBCRL has already been used successfully in the coalfield. Bay strains to a distance of 7.6 metres into the rock can be measured to an accuracy of 0.03 mm (0.001 inches). Installation is relatively simple in pre-drilled holes, and up to 20 reference points (or 21 bays) can be monitored. The reference point locations can be easily set to correspond to the strain bays on monitored rock bolts.

A simple extensometer forms the basis of a tell-tale that can be used to detect dangerous rock movements that may occur during weekends or holidays. A point anchored resin bolt (one fast-set capsule placed at the top of the hole), 0.5 m longer than the regular bolts is installed at regular intervals along the trial section without the nut or plate. Rings of 1 cm wide reflective tape are placed around the bolt at 1 cm intervals

on the exposed steel bar. This will result in three rings of tape being visible, with the upper ring level with the roof, if the bolt is installed correctly.

During pre-shift inspections prior to the entry of workmen, the supervisor can count the visible rings and determine whether or not any large movements have taken place since the last inspection. This type of tell-tale is recommended when the roadway is supported on rock bolts alone as an effective indicator of workplace safety.

5.5.2.6 Load Cells

The loads on the standing support and on the rock bolts themselves should be measured so that the effects of modifications to the support design can be determined. A variety of existing methods are available for determining the load on standing supports, such as glotzl-type pressure cells, oil filled dynamometers and weldable strain gauges mounted on steel legs.

Loads on rock bolts can be obtained in a number of ways. Conventional bolt load cells consist of oil filled dynamometers or electrical strain gauge resistance load cells in cylindrical form that mount on the bolt between the nut and the bearing plate. Simpler devices, such as rubber rings that deform under load, spring washers, or photo-elastic indicators have also been used. Although these devices are of most use when dealing with point anchored, tensioned rock bolts in which load is distributed evenly along the free length of the bolt to the mouth of the hole where it can be measured, the more inexpensive, reusable types may have some application in monitoring initial setting loads.

Full column resin bolts generate local loads at zones of bed separation which would not be detected using conventional bolt load cells. Strain gauged rock bolts, or extensometer bolts provide a more accurate means of determining loads in this type of support.

5.5.2.7 Closure Monitoring

Closure monitoring is carried out by measuring the distance between pins or bolts installed in the roof, sides and floor. Closure should be measured at three points across the roadway so that roof beam buckling or tilt can be detected (Figure 5.4). In an arched roadway, the closure should be measured from wire extensometers of the type described above with anchor locations at or just above the crown level.

5.5.2.8 Monitoring Frequency

Readings should be taken at all measuring points every shift until the effects of extraction are complete. This will take about one or two weeks, depending on conditions, but plotted results will indicate the stabilization point. Daily readings should then be taken until the end of the trial, and at least weekly after that for as long as possible.

Where possible, readings should be taken by the rock bolting team, but supervisors in the bolting trial areas or colliery surveyors can be instructed in reading the instruments as a backup. Forms for recording readings should be included on shift reporting sheets.

5.5.3 Assessment and Adjustment

As the data accumulates, it should be compiled and analyzed. Stabilization of closure and load build up indicates safe conditions, but continued increases in readings should be investigated and remedial measures taken where appropriate. The comparison of support loads in bolted and unbolted sections will allow decisions to be made on the advisability of reducing arch frequencies, and also the need for changes in the bolting pattern.

5.6 Training Programmes

The introduction of rock bolting as a primary means of support at a mine to supplement or replace free-standing supports presents psychological problems for both management and the workforce. The main cause of concern has been found to centre on the perceived strength of roadway supports and the fact that free-standing supports can be observed reaching yield and failure. In the U.K. it has been recognised that, whilst roofbolts can provide improved roadway support they do present a bigger risk in terms of support design and installation standards. In simple terms, free-standing supports possess a larger margin-of-error. It is essential, therefore, that all personnel who will be involved with roofbolting at a mine are fully trained in the technique.

The instructors used in this operator training should be fully versed in all aspects of roofbolting and able to handle detailed technical questions and 'awkward' questions, especially with regard to safety. Use can be made of specialist engineers, technical representatives from major suppliers, outside consultants and trained internal instructors. A variety of speakers is useful as it stimulates interest and reinforces the idea of the importance of the subject. It is vital, however, that the speakers do not contradict each other. For this reason, there is a feeling that one speaker should be used throughout the training exercise, notwithstanding the above advantages. The other speakers, if used, should be under his/her direction.

5.6.1 Form of Training

It is important to explain the principles behind the trial, the methods to be used, and the importance of compliance with design standards. Safety issues must be addressed and resolved, and reasonable questions answered so that everyone involved has confidence in the proposed system. A suitable training program involves the following stages;

- presentation of details of the rock bolting technology to colliery staff and workers to be involved in the trial. This should include an overview of theory, methods, equipment, and the proposed program.

- a visit by colliery representatives to existing sites in coal mines. In this case, such a step may involve a visit overseas. Although the cost would appear to be high, worker acceptance of the technology will be improved, particularly if the visit is undertaken by workmen and officials.

- assembly of all proposed equipment in a mock-up situation (a training gallery underground or on surface), followed by training of personnel by manufacturers representatives on surface.

- transfer of equipment to the underground site, with on-site training and supervision, again by manufacturers representatives, until such time as the workmen are familiar with the equipment and technologies.

- follow-up training and refresher courses to acquaint new or transferred employees with the methods and equipment.

- certificates awarded for completion of training courses.

Initially, the personnel for the roofbolting exercise should be identified after discussions between line management and mine training officials. The selected personnel should be asked if they are willing to take part in the exercise. This step will help to prevent possible disruption during the implementation phase.

Actual training should take place away from the normal workplace in a classroom environment for the theory and in a training gallery or underground site for the 'hands-on' experience. This step helps to reinforce the importance of good practice as an aid to successful roofbolting.

The very best training aids should be used and these should include videos, slides and items of rock bolting hardware. The requirement for visual aids is important. The actual hardware, such as bolts, capsules and machines are necessary so that the operators know exactly what is being discussed and proposed. Whilst the videos and slides will show bolt installation, it is very important that the training phase includes practical exercises during which holes are drilled and bolts are installed in the correct manner. Some of the operators may be familiar with the procedures, but it is probable that they have developed poor or unacceptable techniques. Any such errors can be pointed out at this stage.

5.6.2 Training the Workforce

Training of the work force will centre on correct installation methods and safety. Initially an overview of the trial programme should be given, indicating the staged and conservative approach adopted to ensure the safety of the workplace. A description of the support mechanisms of roofbolting and a comparison with the mechanisms of free-standing supports should also be given. The hardware to be used should be described, with examples such as bolts, capsules, plates, mesh, straps and machines. A description of the installation procedure and a full explanation of the factors affecting bolt efficiency should also be given in the classroom.

In addition to the classroom work, a full practical exercise with the equipment to be used during the trial should be conducted either underground or in a test gallery; conditions should be as close to the proposed underground trial site as possible.

Safety and safe working practise should be emphasized at all times.

5.6.3 Training of Management/Supervisors

All management and supervisors should be just as knowledgeable as the workforce on the basics of roofbolting described above. It has even proved valuable in the U.K. for management involved on a day-to-day basis with underground operations to have at least drilled holes and inserted bolts correctly. However, management also require information in addition to the practical installation and safety aspects; economic factors, geotechnical assessment, bolt testing and instrumentation and monitoring should all be explained.

5.7 Discussion

The following aspects of the detailed programme design have been reviewed above;

- assessment of support requirements and bolt pattern design procedures,
- rock bolting equipment and supplies,
- monitoring and analysis programmes, and
- training programmes.

5.7.1 Support Assessment and Pattern Design

One of the advantages of the Hoek-Brown method of assessing support requirements analytically is that the parameter "s" represents the characteristics of the rock mass, and thus allows tentative correlation between rock mass classification and the shape of the failure envelope. On the other hand, Wilson's method of support analysis has been used successfully in the U.K. for over a decade, and gives acceptable results despite the broad generalizations used in developing the theory. The applicability of the Hoek-Brown failure criterion requires validation before it can be used in this setting.

The review of empirical support assessments included in the work by CANMET (1989) found that the CSIR method generally gave the most conservative estimate of support pressure required to maintain the integrity of the roof. This was also been found to be the case during the examination of support requirement assessment methods in the U.K. (Daws, 1990).

Although the NGI system is widely used in the civil engineering and hardrock mining industry, from which most of the baseline data came, it has never found the same acceptance in the coal mining industry. The CSIR system, which was based to a great extent in the coal industry, seems to be the system of choice. It has been found to give the most conservative estimates of support, which in practise have proved to be effective (Daws, 1990).

The rock bolt pattern design methods reviewed by Aston and Gallant (1988) resulted in support designs that varied considerably for a single case study. In many cases the resulting patterns were impractical; specified bolt lengths or densities were too large or small for practical purposes. This is not surprising since many of the design methods are based on back analysis of data from civil engineering construction

in very different rock types. In other cases, only the most general guidelines were given, leaving the detailed design to experience in the conditions concerned.

Similar difficulties in evaluating designs have been experienced in the U.K. and generally speaking no single design method is used. Instead, several design methods are used as a guide to the pattern design, which is then optimised as experience is gained. Daws (1987), recommended the use of the French Empirical method and the CSIR system of estimating required support pressures for use in the U.K., combined with his equivalent thrust method of pattern design.

The basis of all of the support assessment and pattern design methods is an accurate understanding of the geological and geotechnical conditions at each site. This understanding must be obtained from in-situ investigation, core drilling and testing of rock specimens. This information can then be used as input data, along with mining parameters, for the support assessment and pattern design procedures. Based on the above review of design methods, and on experience primarily in the U.K. as previously described, the recommended procedure for determining support requirements and pattern design is as follows;

- conduct an in-situ geological data collection programme including coring, geomechanical analysis and mechanical properties testing of samples,
- estimate the required support pressure and rock load height from data collected in the stage above,
- produce a preliminary rock bolt pattern design based on the rock load height,
- carry out a closure analysis to predict closure at given support pressures,
- modify bolt pattern design to reduce closure if necessary,
- allow for unfavourable lateral stresses by using additional bolts, and
- specify a rib bolting pattern to prevent extension of the roof beam by rib spalling.

One factor affecting the performance of an underground opening is the stae of stress in the rock. In Australia and the U.K. considerable improvements in rock bolting practise have been attributed to the recognition of in-situ stress conditions and the modification of bolting patterns accordingly. To determine in-situ stresses conventionally a complex, time consuming and expensive field programme is usually implemented. A potential alternative, though more simplistic method, is the Triaxial Aqueous Stress Relief Method. The method has been found to be effective in North America (Parker, 1974), and some investigations are now being undertaken in the South Yorkshire Coalfield in the U.K. (Daws, 1990).

5.7.2 Rock Bolting Equipment and Supplies

Rock bolting equipment and consumable items must be selected and ordered in sufficient quantity for the completion of the trial, and must be present for the training stage. Preferred equipment for the trial, and reasons for selection, is described below.

At Prince and Phalen Mine the compressed air supplies are prone to low pressures and volumes, and may not support the use of more than one drilling machine at a time. For this reason, the best options for drilling are considered to be as follows;

- Lingan Mine
  - hand-held rotary compressed air drills of the Wombat or Super Roo type.
- Phalen Mine
  - rotary compressed air drills as above if air supplies are sufficient, or hand held hydraulic rotary drills of the type supplied by Boart, Edeco or Torque Tension.
- Prince Mine
  - as for Phalen Mine.

Conventional electric, hydraulic or compressed air rotary coal borers should be supplied for the installation of rib bolts.

The bolt type preferred for initial trial in the Sydney coalfield is a 25 mm diameter Grade 400 ribbed reinforcing bar installed as a pre-tensioned unit in 32 - 35 mm. holes by employing fast and slow setting resins in the borehole (Figure 5.5). Smaller diameters of bolt can be used if made of high strength steel to produce similar yield and ultimate loads. The systematic use of alternative bolt types, for example Grade 400 20 mm. resin rebar or "Swellex" friction anchors, should be deferred subject to the analysis of the results of investigations of the trial roadways, and only considered for later installations. Rib bolts should be of either wood or fibreglass, full column grouted, and installed at the same time as the roof bolts.

Drill rods and bits can be ordered from the drill suppliers, but it is usually less expensive to order them from the manufacturers. Drill suppliers should be asked to specify the rod type for their machines and the type and size of bit connector required for water flush drilling. Water flush is essential to prevent dust, reduce bit wear, improve drilling speeds, and produce a clean hole for effective rock/grout bonding.

The preferred rod is a 22 mm. hexagonal rod in lengths appropriate to the rock bolting patterns selected and the roadway and drill parameters, with a central water flushing route.

Drill bits should be negative rake asymmetric two wing bits with 20 mm. screw connections and "tough" grade carbide inserts. Sufficient quantities should be ordered to allow frequent bit changes and rapid drilling rates, or arrangements made for bits to be re-sharpened. After sharpening, a bit gauge should be employed to ensure that the hole diameters will not be less than 32 mm.

Because the outcome of the rock bolting trial is to be the adoption of widespread, systematic rock bolting, hence the purchase of large quantities of equipment, drill manufacturers may supply drills and accessories for the duration of the trial on a demonstration, rental, or sale or return basis. If so, they should be asked to supply rods and bits to match the bolting pattern and available height of the opening, although, as consumable items, these will almost certainly have to be purchased. This has two benefits; the capital

cost of the trial is reduced, and the operational qualities of each machine can be determined prior to purchase.

All bolts should be equipped with torque nuts. If the plastic insert variety is used, some method of preventing fouling of the threads by plastic swarf should be present. Bolt plates can be ordered with the bolts, or fabricated locally. Either wedge washers or dome washers should be used to ensure that angled bolts are tightened flush to the plate and reach maximum tension on installation.

Weldmesh lagging in the form of panels or strips should be provided at all trial sites to replace the current galvanized sheets. In arched roadways, wooden wedges should be used between the weldmesh secured in place by the bolts, and the steel work. In flat-topped roadways, the mesh should be held in place with steel straps pre-bored to indicate rock bolt positions. Straps should also be used to provide restraint between the rib bolts.

5.7.3 Monitoring and Analysis

A full programme of monitoring as described above is essential to the success of the trial. Results from both the technical and geotechnical monitoring programmes will be used to design and optimize the bolt pattern and installation procedures, and to maintain safe working conditions.

The monitoring programme should be implemented by competent, qualified and experienced personnel.

5.7.4 Training

Training of personnel should be initiated well before the underground trials commence. Of special importance is the training of the Trial Supervisors and rock bolting technicians employed by CBDC to oversee the trial. As these people gain experience, they can themselves conduct the required training programmes and refresher courses required.

Both management and workmen should be encouraged to complete the training courses, which should include practical experience installing rock bolts in a training gallery or underground setting similar to the working environment. Management training courses should be more detailed than the workers courses, and include rock mechanics and economic aspects of bolting.

An overseas visit by the engineers entrusted with supervising the trial would be an advantage. The inclusion of workmens safety and health representatives and the involvement of Labour Canada would also be beneficial to the progress of the trial. In view of the obvious similarities in mining method and application of the technology, a visit to the British Coalfields is indicated. Potential sites to visit include the Selby Complex, Collieries in the Western Area and training facilities such as the South Yorkshire Training Gallery at Manvers Colliery.

A suggested syllabus for each of the required training programmes is presented as part of Appendix 2.

6.0 IMPLEMENTATION OF ROCK BOLTING TRIALS

6.1 Review of Investigations

Information gathered during the investigation and documented in the preceding chapters allows the development of an implementation strategy for the proposed rock bolting trials. The first phase of the investigation, the assessment of current and future requirements, exposed CBDC's need for improved driveage rates to support the development of retreat longwall extraction and more economical methods of support to remain competitive. It is anticipated that these requirements will drive the progression of the rock bolting trial from the demonstration of rock bolt technologies through to the use of rock bolts as the sole support in headings driven in suitable conditions.

The second phase of the investigation examined the procedures that must be followed to meet the requirements of the current legislation. These included submission of a programme plan to the CMSC, notification of Workers Safety Representatives at the collieries concerned and provision of training and documentation for the trial. The third phase resulted in the selection of trial sites at each of the three operating collieries in the coalfield, as requested in the contract. The preferred trial sites, and the reasons for their selection, are discussed further below.

The last phase of the investigations was the examination of the rock bolting system, from design of the required support through to training of personnel. This resulted in the selection of design methods, equipment (including drills, bolt type, e.t.c.), and the development of monitoring and training programmes.

6.2 Development of an Implementation Strategy

In order to effect the transition from conventional steel supports to support by rock bolts alone in safety and with the confidence of management, workmen and regulatory agencies, the same stepwise approach adopted in the design of the rock bolting patterns must be followed in the implementation phase, as follows;

- Step 1: Monitor roadway as presently supported.
- Step 2a: Introduce rock bolting between existing standing supports and monitor the effects.
- Step 2b: If required, modify bolting pattern based on results of monitoring Step 2a, and continue monitoring.
- Step 3: Open standing support spacing to 2m. Keep bolt pattern as before and set wooden or lightweight steel supports under the straps, monitoring support and strata behaviour as before.
- Step 4: Support roadway on bolts, straps and wooden/light steel legs only. In the case of gateroads, supplementary support, in the form of temporary hydraulic posts should be available to counter the front abutment effects, if required.

- Step 5: Continue drivage on this pattern if monitoring results are satisfactory.

Based on this sequence of events and those described previously, a comprehensive rock bolting field trial programme leading to the effective use of rock bolts as the sole method of support under favourable geotechnical conditions must consist of the following stages;

- identification/confirmation of trial sites
- preliminary geotechnical investigation
- bolting pattern design
- selection and acquisition of equipment
- notification of CMSC and Workers Representatives
- training of personnel
- stepwise implementation of rock bolting (as described above) and associated technical and geotechnical monitoring to allow assessment and adjustment of the systems and methodologies.
- use of rock bolting as sole support under appropriate conditions in the coalfield.

Full consultation between CBDC, Unions and Labour Canada must be maintained at each stage of the move towards rockbolting as a primary means of support. If at any time the results of the monitoring show an unstable trend, then this will be identified sufficiently early to allow extra support to be set or to revert back a step.

In addition to the above approach, the following comments can be made regarding the implementation of the programme;

- all personnel involved with the rockbolting exercise should have undergone appropriate training.
- a competent person with specialist knowledge of rockbolting should be appointed as co-ordinator of all bolting activities in the coalfield. He should report regularly to appropriate line management and have responsibility for the whole exercise on a daily basis.
- on each shift, a supervisor of engineer status should be appointed to oversee all bolting operations on that shift. In the initial stages no bolt should be installed unsupervised. These engineers should also be responsible for ensuring that monitoring equipment is installed and maintained and for collecting and recording the data.
- no changes to the agreed plan of action should be made without full consultation except where emergency remedial action is required.

a meeting of all interested parties should be held on a weekly basis during the trial to discuss progress and agree future actions.

6.3 Organization and Responsibilities

Some aspects of the organization of the trial have already been referred to. As the users of the technology, CBDC, both at Headquarters Staff and Colliery Level, will be the main participants. However, the smooth progression of the trial also requires the involvement of Unions, Labour Canada and the CMSC. In addition, it is expected that CBDC will ask for the assistance of CANMET-CRL Sydney in the monitoring of the trial. Consulting services from outside CBDC and CANMET will be required for the design of rock bolting patterns, training of management and workmen, and the analysis of monitoring results for system optimization.

The division of responsibilities during the different stages of the trial is reflected in the Implementation Guidelines appended to this report. The guidelines describe a logical, step-wise approach leading to the use of rock bolts as the sole method of support in roadways under favourable conditions.

6.4 Preferred Trial Sites

Whilst potential trial sites at all three operating collieries have been selected, there are strong arguments to be made for further critical review and restriction of the trial to two of the locations previously described.

Geotechnical and mining conditions for the implementation of rock bolting are best at Prince and Phalen Collieries. Preliminary evaluation of rock mass quality and geology at the suggested locations in both mines suggest that rock bolting as the sole means of support has a good chance of success in terms of reducing driveage costs and increasing productivity. Both collieries employ flat topped gateroad driveage methods, which reduces disruption of the roof beam and adds to the stability of any rock bolt support system.

Lingan Colliery, on the other hand, has worse geotechnical and mining conditions, and neither of the two trial locations identified in discussions with colliery staff offer reasonable expectations of success in terms of the widespread use of rock bolting as a sole support. The 13 East Bottom Level is to be driven with an arch profile, which destroys the integrity of the roof. It is to be re-used for the 14 East Panel, subjecting the support system to two abutment stresses. Whilst rock bolting can be used to provide supplementary support to the steel arches, there is little prospect of successful rock bolting as the sole means of support without the need for major repairs to the roadway at a later stage. The Deeps Connector Driveage, the alternate trial location at Lingan Colliery, is to be driven under better geotechnical conditions in terms of depth and effects of adjacent workings, but experience at this location would not be transferrable elsewhere for these very reasons.

The preferred locations for trials implemented to fulfil the intent of this contract are therefore Prince Colliery and Phalen Colliery. Despite the preference shown by Prince Colliery engineers for a trial room for the testing of rock bolting patterns, the Prince trials should be implemented in an active gateroad at a location

likely to cause the least disruption in the event of problems developing i.e. at the beginning or end of the panel. Results obtained from these trials would represent use in a production situation, and the stepwise implementation described would prevent developing problem affecting current productivity levels.

The trials at Phalen Colliery should be implemented behind the new VME Roadheader, i.e. bolts should be installed behind the powered supports used to provide temporary support to the newly exposed roof. The reasons for this are twofold: firstly, there are practical difficulties associated with bolting in front of the driveage machines employed for gateroad development at Phalen Colliery, and secondly, the method of installing the support in the suggested trial heading has to be changed anyway, offering minimized resistance from the workmen to rock bolting as a support method.

7.0 ROCK BOLTING SYSTEM COST STUDY

7.1 Introduction

Detailed cost studies have been prepared for the rock bolting trial. The studies were conducted to achieve the following goals:

- to identify the various cost components associated with the current support techniques,
- to quantify and qualify the various cost components associated with the recommended support techniques, and,
- to determine the costs associated with implementing the recommended technique and reducing the reliance on the current methods.

The calculation of rock bolting costs should be prefaced with the following remarks:

- The cost studies rely on the accuracy and type of data provided by CBDC. In some cases this data did not comply with the requested format, and in other cases was incomplete. The studies have been concluded using the data provided.
- Costs per metre advance depend heavily on the driveage per shift. Although the use of rock bolting as the sole means of support will very likely improve driveage rates, we have made cost estimates on the basis of current driveage rates. Cost benefits attributable to better productivity will be additional to the cost benefits described.
- No cost benefits could be calculated for the better ground control, and hence improved stability, ventilation, repair materials and maintenance costs, brought about by the use of rock bolts. Cost benefits due to these factors will be in addition to those stated.

7.2 Costs of Current Support Technique

Representatives of the engineering departments of Lingan, Phalen, and Prince Collieries were contacted and requested to provide specific information on the costs associated with the support techniques currently used at the test locations. The information requested consisted of the following:

- Projections or estimates of the roof geology.
- Manpower: number, grade, and salaries
- Direct and indirect manpower costs (overtime and benefits, etc.)
- Type, cost and amount of supplies per metre of advance
- Materials support, unit costs, costs per metre of advance

- Machinery type, cost per metre of advance, machine availability in minutes per shift, aqnd times spent in cutting, support erection, and standby.

- Cycle times

Some information received from the collieries was aggregated, i.e. a single labour or materials charge; some of these costs may include utilities and/or repairs that could not be separated. Cost data provided in Canadian Dollars per Foot of Advance ($/Ft.) were converted to Canadian Dollars per metre of Advance ($/m) for uniformity of units and comparison purposes.

Cost data recieved from Prince and Phalen Collieries were for current development costs. The cost data obtained from Lingan Colliery were costs for the 13 E Bottom level. These costs included rock bolt costs. In order to be able to compare costs from all three collieries before and after the installation of rock bolts, costs associated with rock bolting at Lingan were subtracted from the costs obtained from CBDC, and the equivalent conventional drivage costs presented. Details of these and other drivage costs for men and materials received from the Collieries is presented in Table 7.1. These costs exclude the advancement of utilities and repairs as these items will be incurred regardless of the support system used.

Table 7.1 can be summarized as follows;

COLLIERY	MANPOWER	MATERIALS	TOTALS
PRINCE	$207.19	$347.54	$554.73
PHALEN	$357.00	$489.85	$846.85
LINGAN	$694.96	$490.27	$1,185.23

7.3 Costs of Recommended Support Techniques

The costs associated with the implementation of rock bolting can be considered in three stages;

- the capital costs of rock bolting equipment

- costs of consumable items

- manpower costs

For the purpose of the cost study the following bolting parameters have been assumed;

- full column resin tensioned reinforcing bar type rock bolts of 25 mm diameter installed in 32 to 35 mm drill holes,

- rotary compressed air type rock bolting drills and associated equipment,

- straps and mesh lagging to replace current corrugated iron sheets.

7.3.1. Capital Investment Costs

It was assumed that an investment of capital would be required by each of the Collieries for the purpose of buying a limited amount of equipment. Average equipment costs were obtained from several suppliers. The capital investments were amortized over 24 months, the expected life of the equipment, at a fixed interest rate of 16%.

Capital expenditures total $23,000 for the purchase of three pneumatic drills. The monthly capital cost was calculated to be $1,126, including interest or $18.76/shift assuming 60 working shifts a month. Fluctuations in the interest rate or amortization period will affect the capital charge per shift.

7.3.2 Consumables Costs

Consumable costs associated with rock bolting include;

- rock bolts, nuts and washers
- resin cartridges
- drill rods and bits,
- straps, and
- weldmesh.

Representative costs for these items were obtained from several North American and British suppliers. The following costs were used in the analysis.

BOLT LENGTH	COST (including nut, etc)
1.8 m	$8.00
2.1 m	$9.00
2.4 m	$10.00
2.7 m	$11.00
3.0 m	$12.00

Resin costs were $1.75 for each capsule.

The resin capsule cost is based on a capsule dimensions of 27 mm diameter and 300 mm long. For this size of capsule, a 25 mm bolt and a 32 mm hole the resin yield ratio is 1.5, i.e. 300 mm of cartridge will fill 450 mm of hole. Total consumables costs for each bolt length can be calculated by adding bolt cost and resin cost, obtained by dividing the bolt length by the product of the capsule length and the resin yield

factor, rounded to the next whole number. An allowance of $2 per bolt must then be added to cover drill bit and steel replacement costs. Total bolting costs were calculated as follows;

BOLT LENGTH	COST	RESIN COST	TOTAL
1.8 m	$7.00	$7.00	$16.00
2.1 m	$8.00	$8.75	$18.75
2.4 m	$9.00	$9.00	$21.50
2.7 m	$10.00	$10.50	$22.50
3.0 m	$11.00	$12.25	$25.25

In order to enable a standard presentation of costs at a later stage, the bolting costs can be expressed in terms of $/metre. Using the above figures, the range is $8.33/m to $8.95/m. For a conservative cost estimate, the value of $8.95/m has been selected for use later.

In an earlier section of the report the replacement of corrugated steel panels with weldmesh was recommended in conjunction with rockbolting. Current costs for corrugated steel panels are approximately $13/m^2 of coverage, which includes an allowance for overlap of the panels. No supplier of weldmesh for mining use was found, but approximate costs for a similar product in use in the U.K. are $6/m^2 based on an exchange rate of $2.14/UKL. Discussions with the suppliers of these panels indicate that initially they could be supplied from the U.K. at a price competitive with corrugated sheets, and that if sustained use was forseen, they would arrange manufacture in Cape Breton or elsewhere in Canada.

For the purposes of the cost analysis, it was decided to make no adjustment to the driveage costs for weldmesh, despite the fact that it appeared to be less expensive than corrugated sheets because of the uncertainty of supply source and transport costs. Because no costs for sheets were given for Lingan Colliery, an approximate cost of $75.00/metre advance, similar to Phalen Colliery costs, has been added to the materials costs for that colliery.

The cost of providing steel straps between the arches to link individual rock bolts in rows and to act as a lightweight beam when all conventional roof support is withdrawn is based on pre-drilled flat steel strap costs from a Canadian supplier. These costs, for 100mm x 5 mm straps, are approximately $17.50 for a 4.5 m wide roadway. In order to take into account added cost for 'w' section straps, a cost of $30 per row of bolts has been assumed. For bolts on a 0.8 m spacing, the smallest spacing considered practical, this results in a cost of approximately $35/metre advance

7.3.3 Manpower Costs

Manpower costs, $21.50/hour including overheads, for face workers were obtained from CBDC. The manpower costs per shift will depend on whether or not additional men are used to install the bolts, and

how many aditional men are used. A rock bolting crew of two has been found to be the most efficient in the U.K. trials, and two crews can install bolts twice as fast as one crew.

The times required for each rock bolt installation process, obtained from published rock bolting time studies and proprietary information on the files of Jacques, Whitford and Associates Limited, are listed below;

OPERATION	TIME (mins)
Collar Hole	0.25
Drill Hole	0.8 m/min
Insert Resin	2.0
Install Bolt	3.0
Move	2.0

The drilling rate quoted includes an allowance for changing drill steels. Labour charges for typical bolt lengths can be calculated as follows, assuming a two man drill crew;

LENGTH	TIME (manhrs)	COST ($)
1.8 m	.28	$6.09
2.1 m	.29	$6.38
2.4 m	.31	$6.62
2.7 m	.32	$6.90
3.0 m	.33	$7.16

The average time for bolt installation ranges from 4.7 to 3.3 minutes for each metre of bolt, or 0.16 to 0.13 manhours per bolt. The time per metre of bolt decreases dramatically as the bolt length increases from 1.8 m because of the smaller influence of drilling time for short lengths. A value of 4 minutes/metre of bolt, for which the manpower costs are $2.86/m, has been selected as a representative time to install bolts.

7.4 Impact of Rock Bolting on Current Costs

The impact of rock bolting on current support costs depends on a number of factors, some of which are;

- the rate of driveage,
- manpower allocation,

- conventional support densities,
- bolt densities
- variation of conventional support costs with support density,
- time available for installing rock bolts, and
- variation from the costs quoted above.

In order to make even the simplest comparisons, a number of assumptions must be made regarding driveage rates, support density (arches and bolts), time available for bolting and cost variations.

7.4.1 Driveage Rates

To simplify the initial analysis, current driveage rates were assumed, i.e. 7.7 m, 3.7 m, and 2.3 m for Prince, Phalen and Lingan Collieries respectively.

7.4.2 Support Density

Conventional support densities recommended in the implementation of the trial were 100%, 50% and 0% of standard. These have been used in the simplified cost comparisons. Bolt densities have been calculated in terms of metres of bolt per metre advance, according to the following table relating bolt density and bolt length. (a span of 4.5 m is assumed, based on the width of the Phalen and Prince driveages).

4.5 m Span	0.86	1.00	1.12	1.22	Bolt Spacing, (m)
	0.75	1.00	1.25	1.50	Bolt Density (b/m^2)
Lenght (m)	6	5	4	3	Number of Bolts
2.1	12.6	10.5	8.4	6.3	
2.4	14.4	12.0	9.6	7.2	
2.7	16.2	13.5	10.8	8.1	
3.0	18.0	15.0	12.0	9.0	

For the purpose of this study, utilization rates of 6, 9, 12, 15 and 18 m/m have been used.

7.4.3 Time Available for Bolting

Estimates of driveage cycle times were obtained from CBDC for current support methods at all three colliery trial locations. Making the simple assumption that cutting times plus support times minus the available working time gives the 'spare' time available at the face of the heading, i.e. the time available for the installation of bolts with an independent bolting crew, allows the construction of the following simple table:

COLLIERY	PRINCE	PHALEN	LINGAN
Available Working Time	300 min	390 min	390 min
Cutting Time (min/m)	16.7	20.0	78.3
Support Time (min/m)	8.8	45.0	25.0
TOTAL (min/m)	25.5	45.0	103.3
Advance per Shift (m)	7.7	3.7	2.3
Minutes Used	196.5	166.5	237.6
Remaining Time (min/m)			
Current Spacing	13.5	60.4	66.3
50% Current + bolts	17.9	62.9	78.8
Bolts Only	22.3	85.4	91.3

The remaining time, i.e. the time available to install bolts, will restrict the number of bolts that can be installed without affecting driveage rates. The following times are required to install bolt densities of 6, 9, 12, 15, and 18 m/m with 1 and 2 bolting crews;

Bolt Density (m/m)	Number of Crews/Drills	
	1	2
6	24	12
9	36	18
12	48	24
15	60	30
18	72	36

These times are based on 4 minutes per metre of bolt. Maximum bolt densities for each phase of the trial can now be calculated, assuming two bolting crews, for each colliery as follows;

SUPPORT METHOD	MAXIMUM BOLT DENSITY (m/m)		
	Prince	Phalen	Lingan
Current + Bolts	6.7	30.2	33.2
50% + Bolts	8.9	31.4	39.4
Bolts Only	11.1	42.9	45.7

Referring to the table of bolt densities in the previous section, it can be seen that a practical rock bolting pattern installed at Prince Colliery would result in driveage delays in the first instance, but would be marginally practical at the completion of the trial. At Lingan and Phalen Collieries, the low advance rates and reasonably efficient cutting/support cycles allow any reasonable bolt pattern to be installed without hinderance to development rates.

7.4.5 Cost Variations

Disregarding variations in the capital, consumable and manpower costs caused by market forces, materials costs for conventional support driveage will vary as the density of the supports reduces. Data in table 7.1 has been reduced in proportion to the reduction in arch usage during the trial to give driveage costs, assuming manpower costs remain otherwise constant, as follows;

SUPPORT METHOD	MATERIALS COSTS ($/m)		
	Prince	Phalen	Lingan
Current + Bolts	$347.30	$489.85	$490.28
50% + Bolts	$216.62	$308.83	$328.08
Bolts Only	$58.52	$75.76	$165.89

Total costs per metre will be greater than present costs at the start of the trial. As conventional support densities decrease, so the overall cost will alter. The economic viability of rock bolting will then depend on the bolt density required to support the roadway.

7.4.6 Economic Viability of Rock Bolting

The economic viability of rock bolting at each colliery can now be assessed using the data determined above. Currently achieved advance rates are used, and bolt costs of $8.95/m.

The tables given for each Colliery show the saving (loss) for each phase of the trial for bolt densities in metres/metre of advance.

7.5 Summary

Despite the fact that some of the input data was missing or aggregated (see below), the resulting estimation of cost benefits attributable to rock bolting must be considered conservative. It can be seen that, as expected, the implementation of the rock bolting trial will have an adverse effect on the efficiency of operations in the headings, but that this effect will decrease as the density of steel support decreases. In a similar manner, the initial increase in costs will also decrease.

The obvious choice for the initial bolting trial is Phalen colliery, where there is sufficient time to install bolts without reducing driveage rates. However, the broad nature of the assumptions made and their potential effect on the calculation of the economics of rock bolting must be recognised. To assist in further, more detailed calculations, the report includes a suggested spreadsheet programme in Appendix 3. This allows variations to be made to expected costs and the time and cost efficiency of rock bolting operations in the trial headings to be determined.

8.0 CONCLUSIONS

The major conclusions to be drawn from the study described above are as follows:

- The current use of systematic rock bolting within the Cape Breton Development Corporation coal mines is restricted to face salvage work. Economic conditions in the Coal Mining Industry are a major factor in the search for increased productivity and lower costs, and CBDC have expressed considerable interest in systematic rock bolting as the sole means of support of driveages in favourable geotechnical conditions.

- Under the terms of the current Department of Labour Regulations, CBDC are required to develop and implement safe operating procedures for each employee involved in rock bolting operations. A copy of the procedures must be readily available for examination by employees at the mine.

- CBDC are required to submit to the Coal Mine Safety Commission for approval a notice in writing of their intention to carry out the rock bolting trial at least 90 days before it starts. The submission will require pertinent detailed drawings, plans and specifications.

- CBDC are required to prepare a plan of the strata control method including rock bolting for the trial areas and to post that plan at every meeting station in such a manner that it is readily available for inspection by employees.

- CBDC are required to submit plans and procedures in respect in respect of the rockbolting trial to the safety and health committee or the safety and health representative and keep a copy of the approved plans or procedures readily available at the coal mine for examination by the employees.

- Geotechnical conditions at all three coal mines are such that systematic rock bolting offers a possibility of improved safety, support, and productivity. However, the use of systematic rock bolting in arched gateroads at Lingan Colliery as part of a trial to implement total rock bolt support is not practical due to the mining method.

- Phalen and Prince Collieries offer the best opportunities for implementation of roadway support solely on rock bolts.

- Currently available analytical design methods appear adequate for the design of rock bolt patterns, provided that the resulting support is implemented in a controlled and monitored fashion. Compilation of monitoring data obtained during the trial will assist in the optimization of design methods and derivation of coalfield-specific empirical design guidelines for future use.

- Existing rock bolting technology suitable for application in the mining conditions existing in the Sydney coalfield is unmechanised and manpower intensive. Optimization of installation procedures can alleviate some of the installation delays. There are some driveage machines

In limited trial production (Dosco 3000 in seam Miner, and an Australian miner licensed to Anderson Strathclyde) that offer automated cutting and support systems, but these have not been considered for the trial due to their high capital cost.

- Rock bolting hardware suitable for use in the coalfield is available in Canada. Tender calls for the supply of trial materials should include specified properties and dimensions, and all materials obtained should meet these specifications.

- Training of personnel is essential to the success of the trial and should be undertaken by qualified instructors. Both management and workers should be trained in rock bolting, with an emphasis on practical application of the technology. Management should also be trained in the technical, geotechnical and economic aspects.

- A Rock Bolting team consisting of a fully trained engineer with assistance from seconded production supervisors should be selected and made responsible for all aspects of the trial, liaising as appropriate with CBDC Headquarters and Colliery Staff, Labour Canada, Workers Representatives, CANMET and Consultants. This team should hold frequent review meetings with all personnel involved to monitor the progress of the trial and to correct any problems that may develop.

- The rock bolting trial should be implemented in the step-wise fashion described in detail in the report.

- The cost studies indicate that rock bolting offers considerable cost advantages over conventional support when used on its own. When used in conjunction with conventional supports, use of rock bolts increases driveage costs unless arch spacings are reduced or lighter weight steel is used. This conclusion does not take into account cost benefits attributable to improvements in repair costs, strata control, ventilation and safety. In view of the limited information provided by CBDC for reasons of confidentiality, CBDC may wish to conduct their own cost evaluation based on more specific data available to them and taking into account variations in costs of supplies and equipment.

9.0 RECOMMENDATIONS

The main recommendation resulting from the study are is that the rock bolting trial be planned and implemented as described above. Other recommendations and highlights of the implementation guidelines are as follows;

- The recommended locations for the bolting trials are, in preferred order of implementation;
  - 5 East Top Level at Phalen Colliery, or wherever the new VME Miner is to be used,
  - any gateroad driveage at Prince Colliery,
  - the Deeps Connector Driveage at Lingan Colliery
  - The exact locations of the trial sites will depend on the progress of development work and conditions in each roadway.
- Rock bolting installed in the 13 East Bottom level at Lingan Colliery should not be included as part of the current trial. However, it is important that if rock bolts are used, they should be monitored in the same way as suggested for the trial and the data compiled for future use.
- A document detailing the trial and procedures to be followed should be drawn up and presented to Labour Canada, CMSC, Workers Representatives and Colliery management. The document should be submitted to CMSC 90 days prior to the start of rock bolting.
- Training programmes should be developed as suggested in this report and implemented during the 90 day CMSC review period.
- Monitoring of existing support and preliminary design of rock bolting patterns, selection and acquisition of equipment should be undertaken in the 90 day review period. Equipment consumables suppliers should be give detailed information about the site conditions and rock bolting pattern in the tender document.
- Hand held bolting drills of either pneumatic or hydraulic type should be employed depending on the availability and capacity of existing power supplies.
- The recommended bolt type is a 25 mm conventional steel reinforcing bar, equipped with washers, plates and a torque nut and installed in 32 mm. to 35 mm. diameter boreholes.
- Full column resin bonding using fast and slow polyester (i.e. not flame retardant) resins should be employed to allow the bolts to be tensioned at the time of installation. The rock bolting drill should be used to apply as much tension as possible to the bolts. Resin should be selected in consultation with the resin supplier or manufacturer, paying particular attention to ambient temperatures and potential resin losses. Smaller diameter bolts may be used, but only if they have load bearing capacities similar to the 25 mm. diameter bar.

- Bolts should be installed as close to the face as possible, except at Phalen Colliery, where they should be installed immediately behind the powered supports on the VME Roadheader. In rectangular roadways, the side bolts should be inclined towards the solid coal to prevent the formation of shear breaks at the ribsides.

- Each bolt should be installed with a bolt plate to ensure even load bearing on strata at the mouth of the hole and to allow the bolt to be tensioned. The plates may be omitted when bolting through steel straps if the holes in the straps are of small enough size to accommodate the nut.

- In rock bolted roadways the use of conventional tins should be discontinued and replaced with weldmesh panels. Steel sets, where installed, should be wedged between the mesh and the arch.

- In flat topped roadways, the bolts should be installed through pre-drilled steel 'w' straps placed between the mesh and the bolt plate.

- Each bolting trial should be monitored in detail, from both technical and geotechnical standpoints. The monitoring should include unbolted and bolted sections of roadway for comparison purposes. The results of the monitoring program should be continuously reviewed so that adjustments can be made as necessary.

- Monitoring results should be compiled and analyzed to define empirical guidelines for rock bolting in the Sydney Coalfield.

- Research into the development of expert systems for deriving rock bolt patterns and for selecting suitable equipment for individual locations should be undertaken.

- The Triaxial Aqueous Stress Relief Method of rapidly determining in-situ stress regimes should be investigated for possible application to the design of rock bolting patterns in the Sydney Coalfield. The method would allow rapid verification of the insitu stress directions and magnitudes and thus the optimization of the bolting pattern without resorting to expensive conventional overcoring or hydrofracturing methods.

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TABLE 4.1: Summary of Mining Environment at Each Trial Site

Mine	Lingan 13E BL	Phalen 5E Top	Prince
Approximate Depth of Cover	640 m	380 m	230 m
Driveage Method	Roadheader	VME	Dintheader
Shape	Arched	Rectangular	Rectangular
Approximate Driven Height And Width	3.7m High 6.0m Wide	2.8m High 4.5m Wide	2.1m High 4.5m Wide
Support Details	29 kg/m load tread arch 1.2m centres	29.7 kg/m beam on 26.7 kg/m legs 1m centres	4.3m steel beam 2m legs 0.9m centres
Seam (Ht)	Harbour (1.9 m)	Phalen (2.6 m)	Hub (2.1 m)
Compressed Air	Yes (8" line)	Yes * (4" line)	Yes * (low press./vol.)

* *May require additional compressors if compressed air bolting drills are to be used.*

TABLE 4.2

STRESSES MEASURED AT DIFFERENT MINES IN THE SYDNEY COALFIELD

Mine Site	Depth of Test Site (m)[1]	Method of Stress Measurement	Principal Stresses[2] (MPa) (Azimuth/Dip) (Degrees) σ_1	σ_2	σ_3	Stress Components[3] (Mpa) σ_{E-W}	σ_{N-S}	σ_V	Overburden Stress (MPa)
Donkin Mine	165 (12)	overcoring	-	-	-	9.04	7.93	4.90	4.2
No. 26 Colliery	762 (90)	overcoring	26.3 (278/64)	24.05 (9/13)	23.2 (270/-25)	23.31	24.98	25.27	21.7
Prince	222 (20)	overcoring	11.8 (263/14)	9.54 (-7.8/5)	5.1 (64/-75)	10.76	9.51	6.00	5.5
		hydrofracturing	17.8-25.4 (274/0)	12.8-14.1 (6/0)	5.4-5.9 (-)	-	-	-	-
Lingan	593 (70)	overcoring	33.84 (88/4.0)	14.49 (4/-55)	12.21 (355/35)	33.73	13.76	13.05	14.3
		hydrofracturing	-	-	7.8/15.4 (-)	-	-	-	-

Notes:

(1) Depth below sea level, Figure in ( ) indicates depth of water.
(2) σ_1, σ_2, σ_3: Major, intermediate and minor principal stresses.
(3) σ_{E-W}, σ_{N-S}, σ_V: Stress components in E-W, N-S and vertical directions.

From: Canmet, August 1989

TABLE 7.1 Summary of Driveage Costs (excluding utilities) at CBDC Collieries

MINE	ADVANCE PER SHIFT (m)	PERSONNEL			MATERIAL			
		TYPE	NUMBER	COST ($/m)	TYPE	NUMBER	COST ($/m)	
Prince	7.8	Facemen	6	$132.38	Roof Rails	1	$147.60	
		Material	2	$40.90	Legs	2	$91.84	
		Conveyor	1	$20.45	Cor.Sheets	6	$58.52	
		Supervisor	1	$13.46	Wood		$10.93	
					C-Clamps	2	$11.48	
					Struts	7	$27.17	
								TOTAL
		Subtotal	10	$207.19	Subtotal		$347.54	$554.73
Phalen	4.7	Facemen	4	n/a	Arch	1	$345.51	
		Material	2	n/a	Strut	7	$52.08	
		Conveyor	1	n/a	Lagging	5	$8.55	
		Supervisor	1	n/a	Cor.Sheets	8	$75.76	
		Pipe & Mono	1	n/a	Hardwood	15	$7.95	
								TOTAL
		Subtotal	9	$357.00	Subtotal		$489.85	$846.85
Lingan	2.3	Facemen	4	$300.35	Steel Arch	1	$321.89	
		Material	2	$150.17	Hardwood	n/a	$49.58	
		Conveyor	1	$75.08	Wedges	n/a	$2.50	
		Haulage	1	$75.08	Struts	7	$61.30	
		Supervisor	1	$94.28	Cor. Sheets	7	$55.00	
								TOTAL
		Subtotal	9	$694.96	Subtotal		$490.27	$1,185.23

Source: Cape Breton Development Corporation (CBDC), February, 1990.

Table 7.2: Estimated Savings (Losses) Due to Rock Bolting at Prince Colliery ($/m)

	Conventional Support Cost	Bolt Density (m/m) 6	9	12	15	18
Current Support + Bolts	$347.00	($167.03)	($231.89)	($296.65)	($361.61)	($426.47)
50% Current Support + Bolts	$216.62	($36.65)	($101.51)	($166.27)	($231.23)	($296.09)
Bolts Only	$58.52	$121.45	($56.59)	($8.17)	($73.13)	($137.99)

Arches/Steelwork at spacing indicated, including mesh, not tins.

Table 7.3: Estimated Savings (Losses) Due to Rock Bolting at Phalen Colliery ($/m)

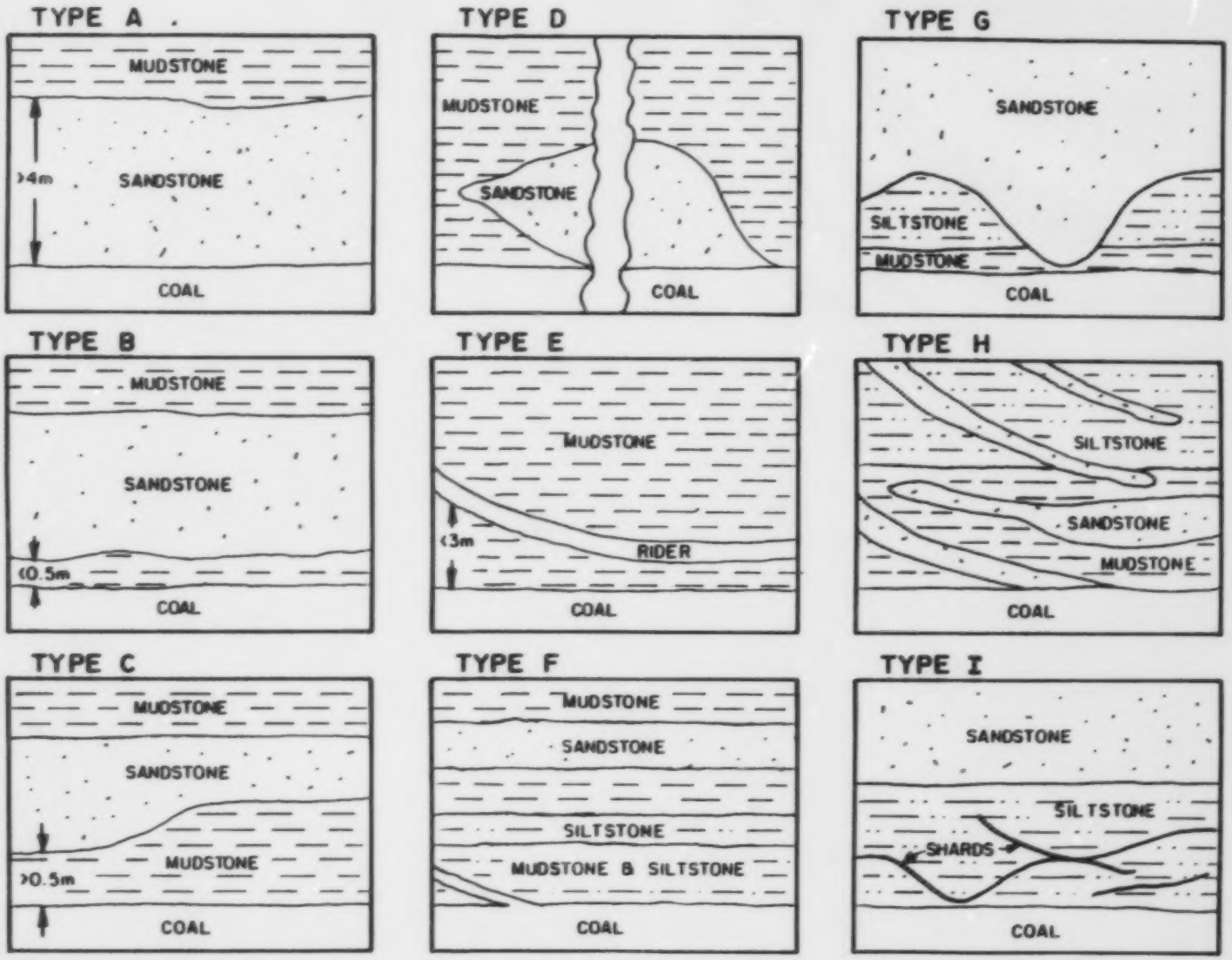
	Conventional Support Cost	Bolt Density (m/m) 6	9	12	15	18
Current Support + Bolts	$489.85	($169.67)	($234.53)	($299.29)	($364.25)	($429.11)
50% Current Support + Bolts	$308.83	$11.35	($53.51)	($118.27)	($183.23)	($248.09)
Bolts Only	$75.76	$244.42	$179.56	$114.80	$49.84	($15.02)

Arches/Steelwork at spacing indicated, including mesh, not tins.

Table 7.4: Estimated Savings (Losses) Due to Rock Bolting at Lingan Colliery ($/m)

	Conventional Support Cost	Bolt Density (m/m) 6	9	12	15	18
Current Support + Bolts	$490.28	($172.75)	($237.55)	($302.35)	($367.15)	($431.95)
50% Current Support + Bolts	$328.08	($10.55)	($75.35)	($140.15)	($205.95)	($269.75)
Bolts Only	$55.00	$262.53	$197.73	$132.93	$68.13	$3.33

Arches/Steelwork at spacing indicated, including mesh, not tins.



ROOF STRATA TYPES OF THE SYDNEY COALFIELD
(CANMET, 1989)

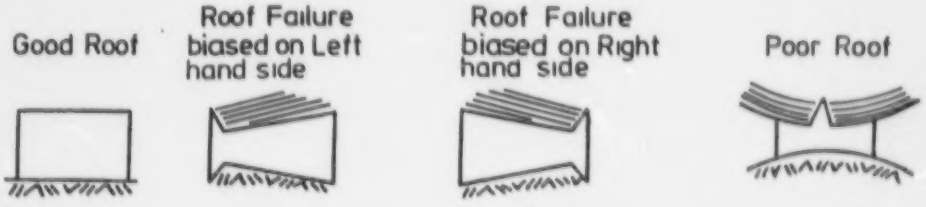
FIGURE 4.1

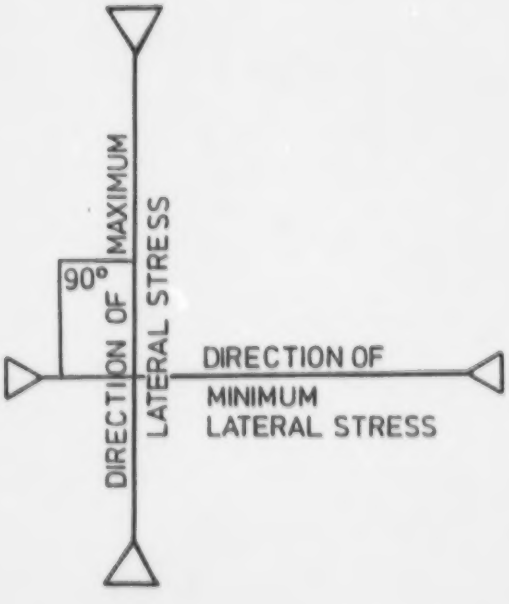
PLAN

Direction of Heading

(^^^^^ INDICATES ROOF FAILURE POSITION)

SECTION (looking inbye)


Good Roof
Roof Failure biased on Left hand side
Roof Failure biased on Right hand side
Poor Roof
Good Floor
Possible Floor Heave on L.H.S
Possible Floor Heave on R.H.S
Excess Floor Heave



ROADWAY ROOF CONDITIONS CAUSED BY THE EFFECTS OF DIRECTIONAL LATERAL STRESS

(AFTER PUGH, 1988)

FIGURE 4.2

SELECTION GRID OF PRINCIPAL PARAMETERS FOR A BOLTING PATTERN

Type of ground	Consequences of ground loading conditions	Bolting parameter						
		Effect of time	Type of anchor	Length	Diameter	Density	Lagging	Remarks
HOMOGENEOUS FEW FRACTURES	In-depth and surface stability	Stabilized deformation						Support useless
		Deferred deformation	Point light	Short	Small	Low	Light	When alterability is high, the layer of sprayed concrete can suffice
	In-depth stability and surface instability	Stabilized deformation	Any	Short	Small	Medium	Light	
		Deferred deformation	Point light or strong	Short	Small	Medium	Light continuous	
	In-depth and surface instability	Stabilized deformation	Point strong or distributed	Medium to long	Medium	High	Heavy	
		Deferred deformation	Point strong	Medium to long	Medium	High	Heavy	
IRREGULAR, LENTICULAR OR FRACTURED IN SEVERAL DIRECTIONS	In-depth and surface stability	Stabilized deformation	Point or distributed	Short	Small	Medium	Light and continuous	Precautionary support
		Deferred deformation	Point or distributed	Short	Small	Medium	Light and continuous	
	In-depth stability and surface instability	Stabilized deformation	Distributed	Medium	Medium	Medium	Light and continuous	
		Deferred deformation	Point, strong or distributed	Medium	Medium	Medium	Heavy	Likely Trial Site Conditions
	In-depth and surface instability	Stabilized deformation	Distributed	Long	Medium	High	Heavy	
		Deferred deformation	Point, strong or distributed	Long	Large	High	Heavy	
STRATIFIED FEW VERTICAL FRACTURES	In-depth and surface stability	Stabilized deformation						Support useless
		Deferred deformation	Point light	Short	Small	Low	Light	When alterability is high, the layer of sprayed concrete can suffice
	In-depth stability and surface instability	Stabilized deformation	Any	Medium	Small	Medium	Light	
		Deferred deformation	Point light or strong	Medium	Medium	Medium	Light continuous	
	In-depth and surface instability	Stabilized deformation	Distributed or point strong	Long	Medium	High	Heavy	
		Deferred deformation	Point strong	Long	Large	High	Heavy	

Notes

(i) Bolt anchors - mechanical or resin point anchors of full column bonded bolts.

(ii) Bolt lengths:-

- Short - 1 m
- Medium - 1/3 opening width (generally between 1.80 m and 2.40 m)
- Long - 1/3 - 1/2 opening width

(iii) Bolt diameter:-

- Small - 20 mm
- Medium - 20-25 mm
- Large - +25 mm

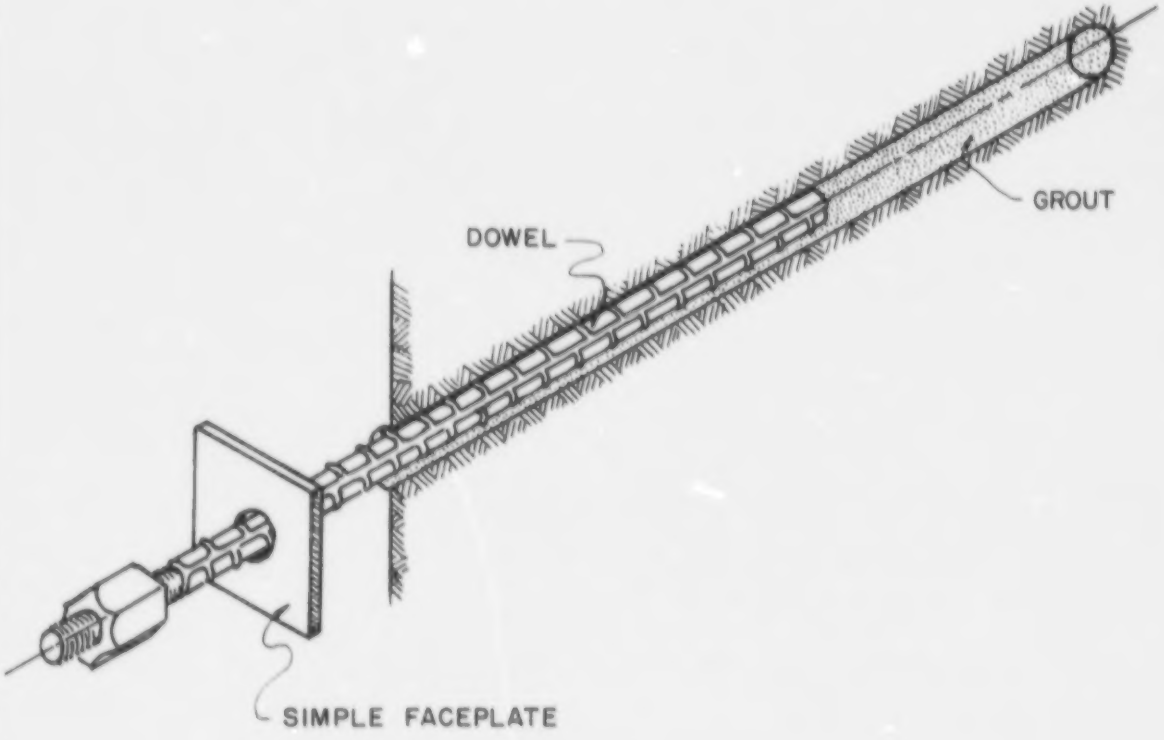
(iv) Bolting density:-

- Low - 0.75 bolts/m^2
- Medium - 0.75 - 1.25 bolts/m^2
- High - 1.25 bolts/m^2

(v) Lagging:-

- Light discontinuous - steel strips or plates
- Light continuous - light gauge wire mesh
- Heavy continuous - heavy gauge wire mesh or weld mesh

Jacques, Whitford and Associates Limited	REFERENCE DEJEAN AND RAFFOUX, 1978	SCALE —— DATE 90/04/07 DWN. BY —— APP'D BY	FRENCH BOLTING GUIDELINES AND DEVIATION OF BOLTING PARAMETERS FOR TRIAL SITE CONDITIONS	DRAWING No. FIGURE 5.1

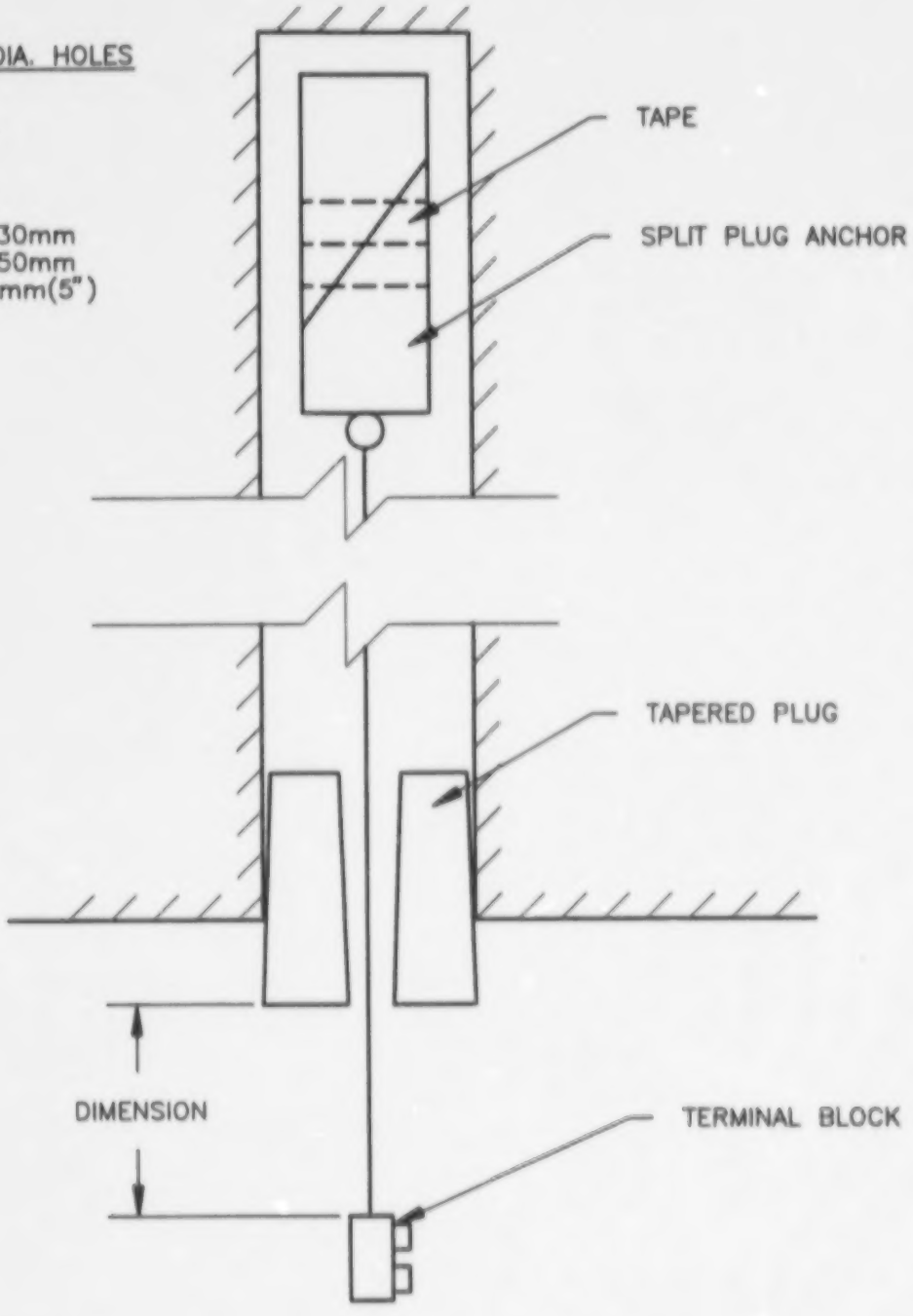


UNTENSIONED GROUTED DOWEL
(TYPICAL, AFTER HOEK & BROWN, 1980)

FIGURE 5.2

FOR 36mm DIA. HOLES
SPLIT PLUG
DIA.: 24mm
SPLIT AT 30°

TAPER PLUG
SMALL DIA.: 30mm
LARGE DIA.: 50mm
LENGTH: 125mm(5")

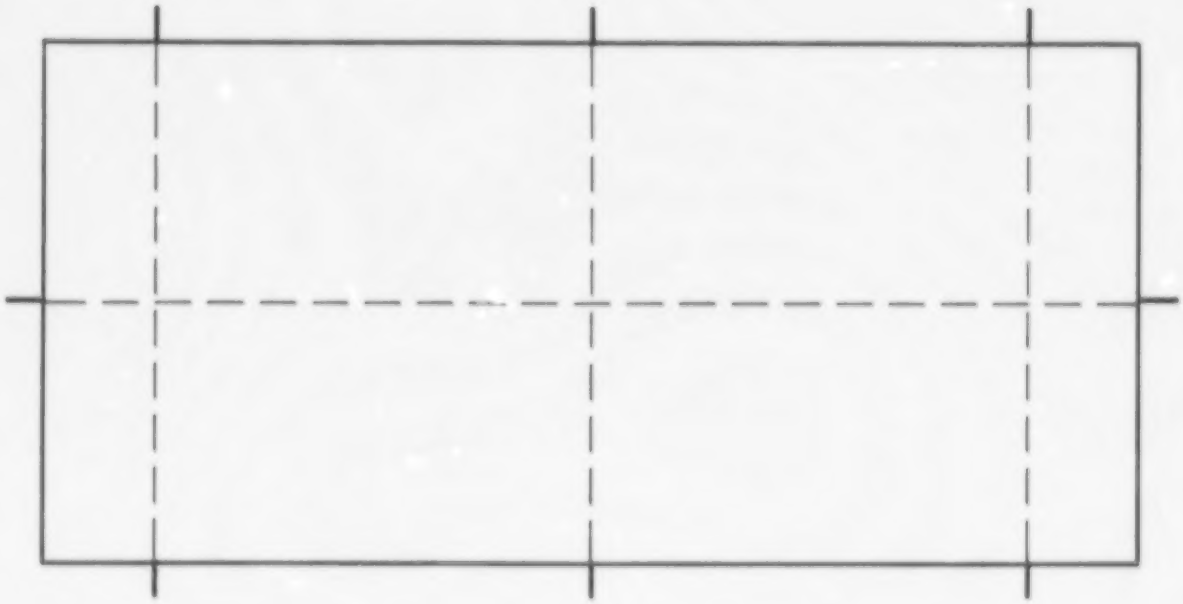


SINGLE WIRE SPLIT PLUG EXTENSOMETER

NTS

240 69/01

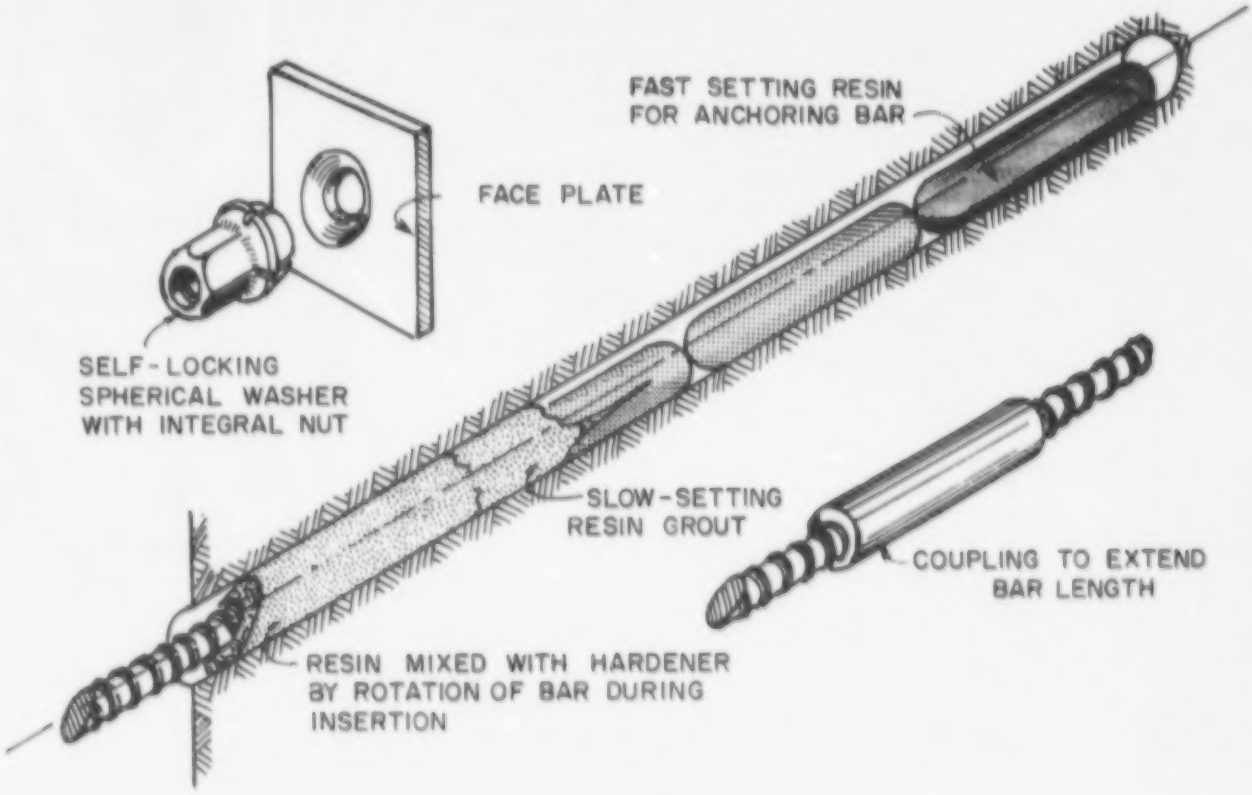
FIGURE 5.3



READING POINTS

MINIMUM RECOMENDED ARRAY OF
CLOSURE POINTS

FIGURE 5.4



TENSIONED, FULLY BONDED RESIN REBAR BOLT
(TYPICAL, AFTER HOEK & BROWN, 1980)

FIGURE 5.5

APPENDIX 1

Design of Trial Patterns: An Example

DESIGN OF TRIAL PATTERNS: AN EXAMPLE

The procedure recommended for the design of the rockbolting patterns for the trial in the Sydney Coalfield was described in Section 5.3.3 of the report, and consists of the following stages:

1. conduct an in-situ geological data collection programme including coring, geomechanical analysis and mechanical properties testing of samples,
2. estimate the required support pressure and rock load height from data collected in the stage above,
3. produce a preliminary rock bolt pattern design based on the rock load height,
4. carry out a closure analysis to predict closure at given support pressures,
5. modify bolt pattern design to reduce closure if necessary,
6. allow for unfavourable lateral stresses by using additional bolts, and
7. specify a rib bolting pattern to prevent extension of the roof beam by rib spalling.

In order to provide an example of the design procedure, details of the 13 East Bottom Level at Lingan Colliery have been compiled. This compilation replaces Stage 1, although it is emphasized here and elsewhere that a full geotechnical investigation should be carried out at each trial site prior to implementation of the trials.

Stage 1: Compilation of Information

Compilation of Information	
Roadway Shape	Arched
Roadway Height	3.7 m
Roadway Width	6.0 m
Support Spacing	1.2 m
Roof Geology	C,F, or G
Rock Mass Rating	45.0
Poissons Ratio	0.25
Youngs Modulus	30,000 MPa
Triaxial Styress Factor	3

Compilation of Information	
Cover Load	20 MPa
Compressive Strength of Rock	50 MPa

Stage 2:Calculation of Rock Load Height and Support Pressure.

Using the RMR value given in Stage 1, the rock load height, h, is calculated as follows:

$$h = \left(\frac{100 - RMR}{100}\right) . S \qquad \text{(A-1)}$$

i.e. h = 3.3 metres.

Given a rock density of 2.7 tonnes/m^3, the rock load per metre of roadway is 3.3 x 6 x 2.7, or 53.46 tonnes, which corresponds to a force of 524 kN or 87 kPa support pressure.

Stage 3: Preliminary Pattern Design

As a starting point in the design process, the French bolting guidelines (illustrated in Figure 5.1), are used to estimate initial design parameters:

- spacing,
  - 0.87 - 1.2 m. on a square grid, or 0.75 - 1.25 bolts per square metre,
- lagging
  - (heavy gauge wire mesh or weld mesh),
- bolt length
  - 0.3 - 0.5 times the span. In this case the span is 6 m, requiring a bolt length of 2 to 3 m. Hoek and Brown (1980) recommend the same bolt length (0.33 - 0.5 times opening width).
- bolt diameter
  - 20 to 25 mm.

The bolt length, L, is determined using the Bieniawski method as h/2 or $S^{0.66}$, whichever is greater (Equations 5-7 and 5-8). In this case, the first form results in a length of 1.65 m, and the second a length of 3.3 m.

The bolt spacing, S_b , is determined from Equation 5-9, as follows:

$$S_b = \left(\frac{C_b}{F \cdot \rho \cdot h}\right)^{0.5} \qquad \text{(A-2)}$$

Substituting values of 1.5, 2.7 tonnes/m^3 and 3.3 m. for F, ρ , and h respectively, and assuming that the anchor capacity determined from in-situ testing exceeds the yield load of the 25 mm reinforcing bar selected as the bolt type, a value of 20.4 tonnes can be substituted for C_b. This gives a value for S_b of 1.23 m.

In calculating the bolt pattern using Daws' method the support pressures estimated from the RMR values (87 kPa) have been used. In practise this would be confirmed by monitoring of the existing standing support loads. The thrust capacity of the standing support system required to support the roof of a 6 m. wide entry with a support pressure of 63.12 kPa is 524 kN/m. In order to support the roadway on rock bolts only, the increase in thrust capacity of the rock arch must match this. Equation 5-10 can be used to calculate the bolt length and spacing:

$$\left(P_a \cdot S_b{}^2\right) + \left(k \cdot C_b \cdot S_b\right) - \left(k \cdot C_b \cdot L\right) = 0 \qquad \text{(A-3)}$$

Adopting a value for q of 3, a bolt yield load of 200 kN and a bolt spacing of 1.2 m., the equation can be solved for L by substituting a value of 524 kN/m for T, giving a recommended bolt length of 2.25 m.

Setting the bolts on 1.2 m centres to match the current arch spacing, the generated support pressure generated by the bolts is given by U/S_b2, or 138 kPa. The required support pressure, calculated in Stage 2, is 63.12 kPa, so the specified bolt pattern should be capable of offering full support to the roadway.

The length of the bolts should be at least 2.25 metres, determined using the Daws method. Bieniawski's method recommended a bolt length of 3.3 m., but this length may cause problems during installation, although the roadway height, 3.7 m., should allow a 3 m. bolt to be installed.

Stage 4: Closure Analysis

Full details of the closure analysis method are described by Wilson (1983), and this work should be referred to for a more detailed explanation. Wilson's equation for determining closure, c, is as follows:

$$c = d \cdot \left(\frac{1+\nu}{E}\right) \cdot \frac{\left(k-1\right) \cdot q + \frac{\sigma}{f}}{\left(k+1\right)} \cdot \left(\frac{2q - \frac{\sigma}{f}}{\left(\rho + \rho'\right) \cdot \left(k+1\right)}\right)^{\left(2+\ell\right)/\left(k-1\right)} \qquad \text{(A-4)}$$

where:
- d = driven diameter (6 m.)
- ν = Poissons Ratio (0.25)
- E = Youngs Modulus of the rock (30,000 MPa)
- k = triaxial stress factor (3)
- q = cover load (20 MPa)

σ = lab determined compressive strength (50 Mpa)
f = lab strength:in-situ strength ratio (4)
p = support pressure (0.063 MPa)
p' = confining pressure offered by rock in yield zone (0.1 MPa)
ξ = expansion factor of broken rock (0.2)

Substituting the above values in the equation given gives a value of 6 cm for the expected closure. The minimum elongation given for 25 mm reinforcing bar is 8%. In a 2.3m bolt, this corresponds to 18 cm. The expected closure is therefore well within the limit of bolt yield.

Stage 5: Modify Pattern to Reduce Closure

As indicated above, the pattern does not require alteration to reduce closure. If alteration had been necessary, then reduction of bolt spacing would result in an increase in support pressure, and hence a reduction in closure.

Stage 6: Allowance for Unfavourable Stress

In-situ stress measurements at Lingan Colliery indicate that the 13 East side of the level will be subjected to high lateral stress, and therefore extra reinforcement should be provided in the form of an extra bolt in the pattern on that side.

Stage 7: Rib Bolting

Rib bolting is required to reinforce the abutment on which the reinforced roof beam rests. Because both coal ribs are to be extracted, steel bolts installed in the sides of the roadway would cause problems in cutting the coal at both face ends. Resin grouted wooden or fibreglass dowels installed in a W pattern, two on each side of the roadway every 1.2 m. are recommended for the initial trial.

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i.e. h = 3.3 metres.

Given a rock density of 2.7 tonnes/m^3, the rock load per metre of roadway is 3.3 x 6 x 2.7, or 53.46 tonnes, which corresponds to a force of 524 kN or 87 kPa support pressure.

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As a starting point in the design process, the French bolting guidelines (illustrated in Figure 5.1), are used to estimate initial design parameters;

- spacing,
  - 0.87 - 1.2 m. on a square grid, or 0.75 - 1.25 bolts per square metre,
- lagging
  - (heavy gauge wire mesh or weld mesh),
- bolt length
  - 0.3 - 0.5 times the span. In this case the span is 6 m, requiring a bolt length of 2 to 3 m. Hoek and Brown (1980) recommend the same bolt length (0.33 - 0.5 times opening width).
- bolt diameter
  - 20 to 25 mm.

The bolt length, L, is determined using the Bieniawski method as h/2 or $S^{0.66}$, whichever is greater (Equations 5-7 and 5-8). In this case, the first form results in a length of 1.65 m, and the second a length of 3.3 m.

The bolt spacing, S_b , is determined from Equation 5-9, as follows;

$$S_b = \left(\frac{C_b}{F \cdot \rho \cdot h}\right)^{0.5} \qquad \text{(A-2)}$$

Substituting values of 1.5, 2.7 tonnes/m^3 and 3.3 m. for F, ρ , and h respectively, and assuming that the anchor capacity determined from in-situ testing exceeds the yield load of the 25 mm reinforcing bar selected as the bolt type, a value of 20.4 tonnes can be substituted for C_b. This gives a value for S_b of 1.23 m.

In calculating the bolt pattern using Daws' method the support pressures estimated from the RMR values (87 kPa) have been used. In practise this would be confirmed by monitoring of the existing standing support loads. The thrust capacity of the standing support system required to support the roof of a 6 m. wide entry with a support pressure of 63.12 kPa is 524 kN/m. In order to support the roadway on rock bolts only, the increase in thrust capacity of the rock arch must match this. Equation 5-10 can be used to calculate the bolt length and spacing;

$$\left(P_a \cdot S_b{}^2\right) + \left(k \cdot C_b \cdot S_b\right) - \left(k \cdot C_b \cdot L\right) = 0 \qquad \text{(A-3)}$$

Adopting a value for q of 3, a bolt yield load of 200 kN and a bolt spacing of 1.2 m., the equation can be solved for L by substituting a value of 524 kN/m for T, giving a recommended bolt length of 2.25 m.

Setting the bolts on 1.2 m centres to match the current arch spacing, the generated support pressure generated by the bolts is given by U/S_b2, or 138 kPa. The required support pressure, calculated in Stage 2, is 63.12 kPa, so the specified bolt pattern should be capable of offering full support to the roadway.

The length of the bolts should be at least 2.25 metres, determined using the Daws method. Bieniawski's method recommended a bolt length of 3.3 m., but this length may cause problems during installation, although the roadway height, 3.7 m., should allow a 3 m. bolt to be installed.

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Full details of the closure analysis method are described by Wilson (1983), and this work should be referred to for a more detailed explanation. Wilson's equation for determining closure, c, is as follows;

$$c = d \cdot \left(\frac{1+\nu}{E}\right) \cdot \frac{\left(k-1\right) \cdot q + \frac{\sigma}{f}}{\left(k+1\right)} \cdot \left(\frac{2q - \frac{\sigma}{f}}{\left(p + p'\right) \cdot \left(k+1\right)}\right)^{\left(2+\xi\right)/\left(k-1\right)} \qquad \text{(A-4)}$$

where:
- d = driven diameter (6 m.)
- ν = Poissons Ratio (0.25)
- E = Youngs Modulus of the rock (30,000 MPa)
- k = triaxial stress factor (3)
- q = cover load (20 MPa)

σ = lab determined compressive strength (50 Mpa)
f = lab strength:in-situ strength ratio (4)
p = support pressure (0.063 MPa)
p' = confining pressure offered by rock in yield zone (0.1 MPa)
ξ = expansion factor of broken rock (0.2)

Substituting the above values in the equation given gives a value of 6 cm for the expected closure. The minimum elongation given for 25 mm reinforcing bar is 8%. In a 2.3m bolt, this corresponds to 18 cm. The expected closure is therefore well within the limit of bolt yield.

Stage 5: Modify Pattern to Reduce Closure

As indicated above, the pattern does not require alteration to reduce closure. If alteration had been necessary, then reduction of bolt spacing would result in an increase in support pressure, and hence a reduction in closure.

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In-situ stress measurements at Lingan Colliery indicate that the 13 East side of the level will be subjected to high lateral stress, and therefore extra reinforcement should be provided in the form of an extra bolt in the pattern on that side.

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APPENDIX 2

Training Programmes

TRAINING PROGRAMMES

Three types of training programme are recommended;

- Workmen training
- Supervisors training
- Management training

Although each type of programme contains similar elements, the level of detail varies and each training programme should be conducted independently. Brief resumes of each programme, duration of session and equipment/facilities required are as follows:

Workmens Training

- Duration:
  - 8 hrs (max 12 participants)
- Requirements:
  - Classroom, VCR/TV, O/H Projector, Slide Projector, Examples of bolding supplies and equipment, Training Gallery, completion certificates.
- Refreshments:
  - Coffee, donuts, sandwich lunch, post-seminar refreshments
- Syllabus:
  - Principles of rock bolting, Objectives of rock bolting - economy, safety, etc., Methods and equipment, correct procedures and their importance for safety, hands on experience in Training Gallery

Supervisors Training

- Duration:
  - 16 hrs (max 12 participants)
- Requirements:
  - Classroom, VCR/TV, OH Projector, Slide Projector, Examples of bolting supplies and equipment, Training Gallery*, completion certificate.

- Refreshments:
  - Coffee, donuts, sandwich lunch, post seminar refreshments (Day 2 only)
- Syllabus:
  - Strata deformation in Coal Mines, Overview of strata control methods and principles, Design parameters, design methods, Principles of rock bolting, objectives of rock bolting - economy, safety, etc., correct procedures and their importance for safety, hands on experience in training gallery, Trouble shooting, monitoring equipment and installation.

Note: 1 day (8hrs) is equivalent to Workers training.

Managers Training

- Duration:
  - 8 hrs (maximum 12 participants)
- Requirements:
  - Classroom, VCR/TV, O/H Projector, Slide Projector, Training Gallery
- Refreshment:
  - Coffee, donuts, sandwich lunch, post-seminar refreshments
- Syllabus:
  - Principles of Rock Bolting, Rock Bolting economics, Monitoring and Quality Control, Correct procedures and their importance for safety.

In order to minimize programme costs, we suggest that the seminars be conducted over as short a period as possible. Five consecutive days would allow 2 workmens courses, a supervisors course and a management course.

APPENDIX 3

Detailed Cost Studies

A B C D E F

CALCULATION OF ROCK BOLTING ECONOMICS

DIMENSIONAL DATA

COLLIERY

Opening Width, (m)	0.00
Bolt Length, (m)	0.00
Bolt Diameter, (mm)	0.00
Hole Length, (mm)	0.00
Hole Diameter, (mm)	0.00
Capsule Length, (mm)	0.00
Capsule Diameter, (mm)	0.00

LEVEL

Arch/Beam Spacing (m)	0.00
Bolt Spacing, (m)	0.00
Bolt Row Spacing, (m)	0.00
Mesh Panel Width, (m)	0.00
Mesh Panel Length, (m)	0.00
** mesh panel size to EXCLUDE **	
** overlap **	

COST DATA

ROCK BOLTING

Total Capital Cost, ($)	$0.00
Bolt Cost, ($)	$0.00
Capsule Cost, ($)	$0.00
Strap Cost, ($)	$0.00
Mesh Cost, ($/Panel)	$0.00
Rod+Bit Costs, ($/bolt)	$0.00
No. Bolting Crews	0.00
No. of Men/Crew	0.00

CURRENT SUPPORT

Steel Cost, ($/Set)	$0.00
Timber Costs, ($/Set)	$0.00
Wedge Costs, ($/Set)	$0.00
Strut Costs, ($/Set)	$0.00
Cost of Tins ($/set)	$0.00
Labour Cost, ($/hr)	$0.00
Paid Hours/Shift	0.00

TIME DATA

ROCK BOLTING

Collar Hole (min)	0.00
Drilling Rate (m/min)	0.00
Place Resin (min)	0.00
Install Bolt (min)	0.00
Move Drill, (min)	0.00

CURRENT SUPPORT

Av. Working Time, (min)	0.00
Time to cut, (min/set)	0.00
Time to Supp., (min/set)	0.00
Delays, (min/set)	0.00

ADDITIONAL DATA

Advance rate (m/shift)	0.00
Resin Wastage, (%)	0.00
Work Days per Month	0.00
Shifts per Day	0.00

A	B	C	D	E	F
DETAILED COST CALCULATIONS					
CAPITAL COST					
Interest Rate, (%)			0		
No. Monthly Pmnts			0		
CAPITAL COST/SHIFT, ($)		=(@PMT($C$20,$C$49/12,$C$50))/$C$56)+($F$41*$F$42)			
OTHER COSTS					
ROCK BOLTING				CURRENT SUPPORT	
Calc. No. Capsules/Bolt		=(((($C$12/2)^2)*@PI)*($C$11*1000)-(((($C$10/2)^2)*@PI)*($C$9*1000)))/((((($C$14/2)^2)*@PI)*$C$13))			
Practical No. Caps/Bolt		@IF(((@INT($C$56)-$C$56)=0),$C$56,(@INT($C$56)+1))			
Calc. No. Bolts/Row		=$C$8/$F$9			
Practical No. Bolts/Row		@IF(((@INT($C$63)-$C$63)=0),$C$63,(@INT($C$63)+1))			
No. Bolts per Shift		($C$41/$F$10)*$C$64			
No. Capsules/Shift		($C$65*$C$61)*(1+($C$42/100))		Steel Support Costs existing spacing, ($/m)	(+$F$20+$F$21+$F$22+$F$23+$F$24)/$F$8
BOLT COSTS, ($/SHIFT)		(($C$66*$C$22)+($C$65*$C$21)+($C$65*$C$25))			
				Projected Spacing with Bolted Support	0
STRAP COSTS, ($/SHIFT)		($C$41/$F$10)*$C$23			
MESH COSTS, ($/SHIFT)		(((($C$8/$F$11)*$C$41/$F$12))*$C$24		Steel Support Costs at projected spacing, with mesh NOT tins, ($/m)	@IF($F$69=0,$C$72/$C$41,(($F$20+$F$21+$F$22+$F$23)*($F$8/$F$69))+$C$71/$C$41)
LABOUR COSTS, ($/SHIFT)		($F$26*$F$27)*$C$26*$C$27			
				CONVENTIONAL SUPPORT COST, ($/SHIFT)	+$F$67*$C$41
BOLTING COSTS/SHIFT		+$C$74+$C$72+$C$70+$C$68			
				PROJECTED SUPPORT COST, ($/SHIFT)	($F$74*$C$41)+($C$70+$C$72)
				SAVINGS (LOSS) DUE TO BOLTING, ($/SHIFT)	-$F$81+$F$78

A	B	C	D	E	F
CALCULATION OF TIME REQUIRED AT CURRENT ADVANCE RATES					
Advance Rate, (m/shift)		+$C$41		Cutting Time, (min/m)	(SF$34/$F$8)
				Support Time, (min/m)	@IF($F$69=0.0,($F$25/$F$69))
Time Per Bolt, (min)		($C$11/$C$34)+$C$33+$C$35+$C$36+$C$37		Delay Time, (min/m)	@IF($F$69=0.0,($F$36/$F$69))
Time per Row, (min)		+$C$90*($C$64/$C$28)		Total Time, (min/m)	@SUM($F$90..$F$68)
Time per Shift (min)		+$C$41/$F$10*$C$91		Time per Shift, (s.m)	+$F$91*$C$68
TOTAL TIME AVAILABLE AFTER CUTTING, SUPPORT AND BOLTING (min)					+$F$33-(+$F$92+$C$92)
					** Note: if negative, cutting, **
					** Support and Bolting cannot **
					** done in time available **
CALCULATION OF ROCK BOLTING COSTS					
PRESENTATION OF RESULTS					
COLLIERY				LEVEL	
CURRENT SUPPORT METHOD					
Steel Spacing, (m)		+$F$8		Advance, (m/shift)	+$C$41
Cost/Set, ($)		+$9,397*$F$8		Cost/Shift, ($)	+$F$87*$C$41
Time/Set, (min)		+$F$34+$F$35+$F$36		Time/Shift (min)	($F$34+$F$35+$F$36)*($C$41/$F$8)
PROPOSED SUPPORT METHOD					
Steel Spacing, (m)		+$F$89			
Bolt Spacing, (m)		+$F$9			
Bolt Row Spacing, (m)		+$F$10			
Bolt Diameter, (mm)		+$C$10		Cost/Shift, ($)	+$F$81
Bolt Length (m)		+$C$9		Time/Shift, (min)	+$F$82
	CONVENTIONAL STEEL CURRENT SUPPORT COSTS, ($/SHIFT)				+$F$78
	PROPOSED SUPPORT COSTS, BOLTS + STEEL, ($/SHIFT)				+$F$81
	SAVINGS (LOSSES) OVER CURRENT METHOD, ($/SHIFT)				+$F$83
	MAXIMUM ADVANCE RATE PER SHIFT, (m)				+$F$33/$F$91+($C$91/$F$10)